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MAY, 1915



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

Mr. _____
(Name in Full)

Occupation _____

Address _____

is hereby proposed by the undersigned, as a _____

of the American Institute of Mining Engineers.

_____ } Signatures of three
Members or
Associates.

Place of birth _____ *Year of birth* _____

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

<i>Dates</i>	
_____	_____
_____	_____
_____	_____
_____	_____
_____	_____
_____	_____
_____	_____

[illegible]

*Present position*_____

_____*Signature.*

Dated _____ *191*

ARTICLE II.—MEMBERS.

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 101

MAY

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUTENON, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

VISIT OF SECRETARY TO LOCAL SECTIONS

The Secretary of the Institute has been appointed as a member of the International Jury of Awards on Mining, Metallurgy, and Geology for the Panama-Pacific International Exposition, and will leave for San Francisco on April 29, stopping off at various places *en route* in both directions to meet members of the Institute. The following itinerary has been arranged, which will be adhered to as closely as possible, although slight changes may be made:

Chicago.....	April 30
St. Louis.....	May 1
Butte.....	May 4
Spokane.....	May 5
Seattle.....	May 7
San Francisco.....	May 10 to 20
Los Angeles.....	May 22
Bisbee.....	May 25
Denver.....	May 29

BOARD OF DIRECTORS

Meeting of Mar. 26, 1915.—The following Committee on Nominations was appointed: Fred W. Bradley, *Chairman*; James F. Kemp, Past President; Frank M. Smith, Chairman Montana Section; R. C. Gemmell, Chairman Utah Section; Frank A. Ross, Chairman Columbia Section; Louis S. Cates, Arthur Thacher.

The Committee on Precious and Base Metals and the Committee on Coal and Coke were appointed, the personnel of which is shown in another part of this *Bulletin*.

The President was authorized to appoint a Committee on Welfare with J. Parke Channing as Chairman.

W. P. Cutter, Librarian of the United Engineering Societies' Library, was appointed a representative of the Institute on the Committee on Classification of Engineering Knowledge to co-operate with similar committees of other engineering societies.

A vote of thanks was extended to Edward B. Sturgis for his excellent work in arranging for the Annual Dinner in February of this year.

The sum of \$57.30 was appropriated to the New York Section.

It was resolved that in response to the courteous invitation from 36 members of the Institute in Arizona and the neighborhood, the 1916 summer meeting of the Institute be held in Arizona.

Resolved, that in the event of any bill for the licensing of engineers being presented to the Legislature of any State, the said Legislature be petitioned to hold public hearings on the subject before such bill is acted upon.

Resolved, that the sum of \$1,250 be appropriated to the International Engineering Congress.

Upon recommendation of the Committee on Membership, 37 Members and 13 Junior Members were elected; and one Junior Member was granted change of status to Member.

The resignations of 20 members were accepted.

The applications of two members for reinstatement were accepted.

SAN FRANCISCO MEETING

The 111th meeting of the Institute for the presentation and discussion of technical papers will be held in San Francisco, Cal., Sept. 16 to 18, 1915. A special train for members of this Institute and for the members of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the Society of Naval Architects and Marine Engineers, and the International Engineering Congress, will be run from New York City to San Francisco, leaving New York on Sept. 9.

All papers to be presented at this meeting of the Institute must be in the hands of the Secretary on or before June 1, 1915.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Mar. 10 to Apr. 10, 1915:

Oscar Lachmund, Greenwood, B. C.
S. W. Ough, Pachuca, Mexico.
B. W. Vallat, Detroit, Mich.
Charles Mentzel, New York, N. Y.
M. K. Rodgers, San Francisco, Cal.

Harvey B. Small, Mangaugue, Colombia.
M. F. Sayre, Schenectady, N. Y.
James W. Hambleton, Parral, Mexico.
Philip L. Gill, Woodmere, L. I., N. Y.

John T. Fuller is President of the F. Shepherd Mining Co., operating at Rush Mountain, Ark.

Frederick Brulé, who was formerly superintendent of the surface department at the Washoe Reduction Works, Anaconda, has been made superintendent of the surface department of the Boston and Montana Reduction Works.

Edward H. Hamilton has resigned from the Norfolk Smelting Co., owned by Beers, Sondheimer & Co., and is now retained by Messrs. Eustis of Boston.

John Palmer Cosgro, formerly general manager in Mexico of the Mine & Smelter Supply Co., has accepted the position of Chief Engineer and General Manager of Sales for the Wilfley Machinery Co., Ltd., with headquarters at Salisbury House, London, E. C., England.

Charles Mentzel left New York in April on a trip of exploration in the Republic of Panama. He will take with him a party of about 50 men.

James Aston, metallurgist, connected with the University of Cincinnati, has resigned his position to enter the service of the U. S. Bureau of Mines to investigate safety and efficiency in steel-plant operations.

Horace V. Winchell, of Minneapolis, and **James R. Finlay**, of New York, have been retained as mining experts for the Jumbo Extension Mining Co., of Goldfield, Nev., in the apex litigation instituted against it by the reorganized Booth Mining Co., of Goldfield.

Utley Wedge has been elected President of the Tennessee Copper Co.

William Campbell has been promoted from Associate to Professor of Metallurgy at Columbia University.

William L. Saunders and **I. C. White** have been appointed members of the Federal Trade Committee of the Chamber of Commerce of the United States, which is to co-operate with the Federal Trade Commission.

Earl S. Bardwell is in charge of the Anaconda Copper Mining Co.'s exhibit at the Panama-Pacific International Exposition.

Claude Ferguson has been appointed Superintendent of the Jesse Mine for the Chaparal Mining Co., Chaparal, Ariz.

W. L. Wright, formerly Vice-President of the Taylor-Wharton Iron & Steel Co., has been elected a director and Vice-President of the Keystone Watch Case Co., Philadelphia.

Charles T. Kruse has been made superintendent of the Marquette, Menominee Range, mines of the Jones & Laughlin Steel Co.

Richard Peters, Jr., who has been connected with the W. J. Rainey Co. interests at Uniontown, Pa., has been appointed assistant to the General Manager of W. J. Rainey Co., with headquarters at 52 Vanderbilt Avenue, New York, N. Y.

Arthur Neustaedter has accepted the position of Superintendent of the Arminius Chemical Co., Mineral, Va.

G. H. Carnahan has resigned as manager of the Teziutlan Copper Co. and Montezuma Lead Co., to become Vice-President of the Intercontinental Rubber Co., 17 Battery Place, New York, N. Y.

B. W. Vallat returned the latter part of March from a three months' trip in Europe.

William Russell, European representative of the Dorr Cyanide Machinery Co., is making a tour of the mining camps of the United States and Canada.

The Dorr Cyanide Machinery Co. announce that after May 1, 1915, their office will be located at Room 2843, Whitehall Building, New York, N. Y., instead of 30 Church Street, as heretofore.

R. H. Postlethwaite is in charge of the Loftus Blue Lead Mines Co., Plumas County, Cal.

T. A. Rickard was the guest of honor at a dinner given by the Institution of Mining and Metallurgy in London on the evening of March 3, on the eve of his departure for San Francisco to resume the editorship of the *Mining Press*. The speeches of the evening gave evidence of the high regard in which Mr. Rickard is held by the members of the mining profession, tribute being paid to him for his services "in advancing the interests and promoting the solidarity of the profession through the pages of the *Mining Magazine* and in other ways."

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, aged 38, 14 years' experience in engineering, exploration, and management of copper, pyrite, and large iron mines, safety work, professor of mining in large university. Married. Open for position after June 10. No. 205.

Member, graduate engineer, aged 44, with over 20 years' experience in connection with large coal and coke operations in different parts of the country, desires position as general superintendent of large coal company, or will take an interest with the management in a smaller company. No. 206.

Member, technical graduate, with experience in testing and operation of blast-furnace, iron-ore concentration and coke-oven plants, and in the examination of low-grade iron-ore properties, desires position with small iron and steel company, in executive capacity. No. 207.

Member, with wide experience in open-pit and underground iron mining, as mine engineer, underground efficiency engineer, superintendent of exploration work, engineer on concentration of iron ores, and as superintendent of mines, will be open for engagement during April, 1915, owing to suspension of mining activities by present company. Best of references given. No. 208.

Member, technical graduate, at present employed as assistant superintendent by one of the world's largest copper companies, desires change. Has had exceptional experience with basic-lined converters, blast furnaces, reverberatories, and roasters; has also had experience in leaching and mill work. Can furnish list of references. No. 209.

Member, metallurgist and mechanical engineer, 14 years' practical experience designing, building, and operating metallurgical works (copper and silver-lead smelters, ore-dressing plants, cyanide mills), specialty metallurgy of copper and lead, open for engagement as engineer, superintendent, or works manager. Has filled responsible executive positions. Speaks Spanish, German, and French. No. 210.

Member, technical graduate, aged 33, nine years' experience as surveyor, assayer, engineer, mine foreman, and superintendent of copper and silver mines in United States, Central and South America, open for position as superintendent at any mine where altitude is under 10,000 ft. Speaks Spanish. No. 211.

Junior member, recent graduate, with some experience in steel plants and as draftsman, chemist, machinist, and furnace foreman. No. 212.

Graduate chemist and metallurgist, aged 27, single, speaks Spanish, German, and Russian, desires position as foreign representative, either technical or business. Four years' experience as chemist and metallurgist; has traveled extensively; good business man; excellent references. Available June 1. No. 213.

Member, with technical education and 20 years' experience in mining, milling, and cyanidation in Colorado and Canada. For several years superintendent of a large Colorado mine. Has examined and reported on mines and prospects in all the Western States, in Canada, and Mexico. Has himself performed nearly every kind of work in mines and mills. Also had charge of hydro-electric plant construction. Willing to go anywhere. No. 214

Member, technical graduate, married, aged 32 years, ten years' experience as machinist, surveyor, miner, mine and mill foreman, mine and mill superintendent, in operation and construction. References. No. 215.

Junior member, technical graduate, two years' experience in chemistry and general engineering, would like a position in mining or metallurgical work. No. 216.



SAN FRANCISCO LOCAL SECTION

*Executive Committee*G. HOWELL CLEVINGER, *Chairman*C. W. MERRILL, *Vice-Chairman*JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.

F. W. BRADLEY

ANDREW C. LAWSON

Meets on the second Tuesday of each month

Farewell Dinner to H. Foster Bain

H. Foster Bain, who has gone to London to assume the editorship of the *Mining Magazine*, was tendered a farewell dinner by the San Francisco Sections of the American Institute of Mining Engineers, the Mining and Metallurgical Society of America, and the Engineers' Club, on the evening of Mar. 9, 1915, at the Hotel Bellevue, San Francisco, 85 members being present.

George H. Clevenger acted as Toastmaster and the following toasts were responded to:

Bain and the Mining Industry, E. H. Benjamin.

Bain, the Geologist, A. C. Lawson.

Bain, the Man, F. W. Bradley.

Bain, the Friend, C. W. Merrill.

Bain, the Schoolboy, C. F. Tolman.

The Frontispiece and Flow sheet from the souvenir program, designed by James C. Ray, Secretary of the Section, are reproduced herewith.

FLOW SHEET

Slime Ore.....	(Oyster Cocktail, Club Style).....	to washer	{ Ore—bins Slime—waste
Stock Solution..	(Cream of Asparagus, Crouton Soufflé).....	battery feed—	agitators
Flint Pebbles.....	(Ripe Olives).....	tube mills	
Lowgrade Ore....	(Medallion of Sea Bass, etc.)....	hand sorting	{ Ore—rolls—wilfleys tails—dump
Highgrade Ore....	(Tenderloin of Beef, Larded, etc.)....	{ jaw crushers—stamps— classifiers—regrinding— agitators—Merrill filters	
Froth.....	(Punch Chartreuse).....	from Pachucas	
Specimen Ore....	(Stuffed Squab, Chicken, etc.)....	panamalgamation—	retorts
Free milling Ore.....	(Hearts of Lettuce with Egg, etc.).....	stamps—plates	
Bull quartz.....	(Vanilla Ice Cream).....	to waste	
Slime cake.....	(Cheese).....	from Oliver filter	
Pregnant solution.....	(Coffee).....	zinc boxes, sump	
Strong Solution*.....	(Wine).....	head tanks	

* First aid outfits for cyanide poisoning in readiness if necessary.

At the meeting following the dinner W. H. Shockley gave a delightful account of a voyage made in 1910 from Vladivostock to East Cape, northeastern Siberia, for the purpose of prospecting the Siberian coast

opposite Cape Nome. Subsequently, mining claims were examined near Gejiga at the northern extremity of the Okhotsk Sea. Lantern slides were shown illustrating the habits of the Chukchee, Koryasks, Kamchadals, Giliaks and other native tribes of these regions. Mr. Shockley's talk was interspersed with droll remarks.

H. L. Huston gave a talk on Transportation by Camel in Southwestern Siberia, a region little known even to the Russians themselves. Interesting lantern slides were shown, illustrating the various uses to which camels are put, how they are raised, etc.

T. N. Turner followed with a description of some of the novel devices, such as ice lanterns, used in this far away corner of the world

JAMES C. RAY, *Secretary*.

COLUMBIA LOCAL SECTION

Executive Committee

FRANK A. ROSS, *Chairman*

RUSH J. WHITE, *Vice-Chairman*

L. K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.

FREDERIC KEFFER

FRANCIS A. THOMSON

Holds four Sessions during year

Annual Meeting in September or October

As a part of the technical program of the Northwest Mining Convention, which was held in Spokane, Wash., Feb. 22 to 27, 1915, the Columbia Local Section of the Institute held a regular quarterly meeting on the afternoon of Feb. 24, with about 150 members and guests present.

Frank A. Ross, Chairman, presided, and the following papers were presented:

Mine Models, by F. W. Callaway, Kellogg, Idaho, was read by Roy H. Clarke in the absence of the author.

Economic Geology of Northeastern Washington, by Prof. F. M. Handy, was read by the author and discussed by Roy H. Clarke, J. C. Haas, Prof. D. C. Livingston, W. H. Linney, Prof. F. A. Thomson, and others.

Some ideas advanced by the author are not wholly in accord with the views of other geologists who have visited the region and it is expected that the paper will bring out some very lively discussion.

A paper by Prof. D. C. Livingston, entitled Mine Surveying, was reserved for reading at a future meeting.

The convention with technical program in the hands of the several societies of engineers developed much greater interest than would be possible under other circumstances and it is to be hoped that the mid-winter meetings hereafter will be joint meetings and of a similar character, and if the Northwest Mining Convention is to be an annual affair this will doubtless be done.

L. K. ARMSTRONG, *Secretary*.

BOSTON LOCAL SECTION*Executive Committee*HENRY L. SMYTH, *Chairman*ALFRED C. LANE, *Vice-Chairman*HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.

ROBERT H. RICHARDS

ALBERT SAUVEUR

Meets on the first Monday evening of each winter month

The annual meeting of the Boston Local Section, being the 26th meeting, was held at the University Club, Boston, Monday evening, Mar. 1, 1915; 13 members were present.

The Vice-Chairman, Alfred C. Lane, presided at the dinner and meeting. The following officers were elected for the ensuing year: Chairman, Henry L. Smyth; Vice-Chairman, Alfred C. Lane; Secretary-Treasurer, H. A. Wentworth.

The Treasurer presented a report showing a deficit of about \$110, and he was instructed to send a copy of this financial statement to all members of the Section, asking for subscriptions to make up the deficiency.

AUGUSTUS H. EUSTIS, *Secretary*.

UNITED ENGINEERING SOCIETY

To the Board of Directors of the
American Institute of Mining Engineers.
Gentlemen:

The undersigned representatives of the Institute on the Board of Trustees of United Engineering Society make the following report on the recent activities of United Engineering Society, as required by the By-laws of the Institute:

The Board has had under consideration and has referred to a special committee the broad question of policy with respect to memorial tablets in the building.

Bonds representing \$18,000 have been bought for investment.

The Treasurer has received \$5,000 to complete the payments of the American Institute of Mining Engineers in] their share of the land indebtedness.

The Finance Committee has secured the appointment of a salaried accountant and cashier.

The agreement covering plans for financing the joint libraries of the Founder Societies has been accepted and signed in quadruplicate.

The revision of the assessment rate for associate societies in the building has been completed and a slight increase of income from the office floors is to result therefrom.

The Engineering Foundation Board has been created, to consist of Messrs. Albert R. Ledoux and B. B. Thayer of the Mining Engineers, Michael I. Pupin and Charles E. Scribner of the Electrical Engineers, Alexander C. Humphreys and Jesse M. Smith of the Mechanical Engi-

neers, Waldo Smith and Charles Warren Hunt of the Civil Engineers, Edward D. Adams, and Howard Elliott, and Gano Dunn, President of United Engineering Society, *ex officio*.

In February, the Engineering Foundation Board received \$200,000 worth of New York City corporate stock, the income of which is to be devoted to the Engineering Foundation.

On Jan. 26 and 27, 1915, official exercises in connection with the inauguration of the Engineering Foundation were held.

The By-laws of the United Engineering Society have been modified to provide for the receipt and direction of the finances of the Engineering Foundation.

Respectfully submitted,

CHARLES F. RAND.

B. B. THAYER.

JOSEPH STRUTHERS.

AFFILIATED STUDENT SOCIETY NOTES

The University of Kansas Student Auxiliary holds meetings every Wednesday to consider subjects related to mining and mining geology. The meetings are well attended. Good speakers have been secured, as shown by the following schedule prepared for the school year 1914-1915:

Professor E. Haworth: Geology of Kansas (three talks).

Professor Twenhofel: Geology of Baltic Russia.

Geology of Baltic Scandinavia.

Some Parts of the Geology of the British Isles.

Professor Barrell (of Yale): Historical Geology of Connecticut.

Professor E. S. Dickinson: Geology of Lake Superior Iron Regions.

The Use and Practice of the Divining Rod.

Shaft Sinking in the Lake Superior Region.

Professor Todd: The Oil Fields of Wyoming.

Professor Whitaker: Economical subject.

W. J. Squires, K. U., '96: Electrical Shot Firing.

C. J. Hainbach, K. U., '13: Lucky Tiger Mining Practice.

George Belchic, Penn. State, '13: Wisconsin Magnetic Survey.

Clay Roberts, K. U., '13: Copper Mining in Chile.

O. A. Dingman, K. U., '14: Gold Mining in Honduras.

W. E. Rohrer, K. U., '15: Coal Mining in Henrietta, Okla.

G. L. Allen, K. U., '15: Bisbee Quadrangle, Bisbee, Ariz.

C. B. Carpenter, K. U., '15: Electrical Shot Firing in Southeastern Kansas.

The officers of the Geological and Mining Society of Stanford University, California, are: President, C. M. Vrang; Secretary, D. P. Carlton.

BILL MCGINTY ON THE SAFETY MOVEMENT

Secretary, New York Section,
Am. Inst. Min. Eng.

Dear Sir:

Owing to the necessity of attendance at an unofficial meeting of the committee for the investigation of the Public Servants' Omission, Mr. Bill McGinty was unable to be present at the meeting of the New York Section of the Mining Engineers.

He has requested me to submit an outline of his paper, following the custom of the Académie des Sciences, of Paris.

Very truly yours,

T. MONTMORENCY DILLON, I.C.S.

(Indispensable Corresponding Secretary)

Felly-mimbers, ingineers, an' gintlemen:

Havin' had occayshun to invistigate the meltin' point av wires in the con-do-its av the subway, I have consulted manny authorities. I have diskivered that wan interestin' an' purely indickative mode is effected by the use av pin-head particles.

So I sez to Kathleen, sez I, "Give me some av yer pins." Sez she, "Are ye still worryin' about that Safety-First movemint?"

Not havin' anny av the other kind, she helped me to get a supply at the nearest departmint store. There was three kinds; five cints, sivin cints, an' tin cints. Discardin' the five cint kind as unimportant, I conducted me investigayshuns on the more xpinsive kinds.

The appyraytus may be briefly described as a binch vise with sivin inch jaws for holdin' the pins; a thin layer av ciment to protiect a wire carryin' what an ign'rant Mexican wud call *perjuicio de amparo*, which is juice av high ampereage; a steel drill suspinded by the thongs av a black-jack; an' a 12 pound sledge. The operayshun is performed by gently swingin' the sledge against wan ind av the drill. The defeckshun av the other end av the drill is governed by the pin-head; the degree av yieldin' bein' measured by the raysults. Wan pin was used for each experrymint; but in no case did the pin-head serve to protiect the amperes. In spite av the uncertainty av the tests, the whole proceedin' was quite interestin'. I rayported to Kathleen that the pin-heads was no good; but she said, "how about the five cint kind. Thim is the best quality, but they come fewer in the package." Well, I niver thought av that; but we hope to repeat the experrymints some day whin we're waitin' for the naturalizayshun courts to supply us with some good illigible workmen.

Annyhow, I'm glad the nickel pins are the best. Please advise me whin the committee is ready to give me the third degree.

Rayspictfully submitted,

BILL MCGINTY.

References

1907, Vol. 3, pp. 345-355.
14, Vol. 10, p. 77.

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LIBRARY**AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS****AMERICAN SOCIETY OF MECHANICAL ENGINEERS****AMERICAN INSTITUTE OF MINING ENGINEERS****UNITED ENGINEERING SOCIETY****WILLIAM P. CUTTER, Librarian**

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS**PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining and Metallurgy****PRACTICAL WELL SINKING.** By B. A. HARRISON. London, 1913.**WHAT A MINER CAN DO TO PREVENT EXPLOSIONS OF GAS AND OF COAL DUST.** Miners' Circular No. 21, Bureau of Mines. Washington, 1915.**PRAKTISCHES HANDBUCH DER FABRIKATION UND BEARBEITUNG DES STAHL.** By Dameime. Leipzig, 1839.**PRINCIPLES OF METALLURGY.** Ed. 2. By A. H. HIGGINS. London, 1914.**METALLURGICAL SMOKE.** Bull. No. 84, Bureau of Mines. Washington, 1915.**Geology, Mineralogy and Mineral Production****CANADA. PRELIMINARY REPORT ON THE MINERAL PRODUCTION OF CANADA, 1914.** Ottawa, Mines Department, 1915.

- CANADA. ECONOMIC MINERALS AND MINING INDUSTRIES. Panama-Pacific Edition. Ottawa, 1914.
- IDAHO. ANNUAL REPORT OF THE MINE INSPECTOR ON THE MINING INDUSTRY, 16TH, 1914. Boise, 1914. (Gift of State Inspector of Mines.)
- NEW JERSEY. MINERAL INDUSTRY FOR 1913. Bull. No. 15, Geological Survey of New Jersey. Union Hill, 1914.
- MINING DISTRICTS OF THE WESTERN UNITED STATES. Bull. No. 507, U. S. Geological Survey. Washington, 1912.
- MISSOURI. BIENNIAL REPORT OF THE STATE GEOLOGIST, 1915. Jefferson City, Bureau of Geology and Mines, 1915.
- MISSOURI. BASE MAP. 1914.
- NOVA SCOTIA. ANNUAL REPORT OF THE MINES, 1914. Halifax, Department of Public Works and Mines, 1915.
- VERMONT. REPORT OF THE STATE GEOLOGIST ON THE MINERAL INDUSTRIES AND GEOLOGY, 1913-14. Burlington, 1914.
- WEST VIRGINIA. COUNTY REPORTS, 1914. Logan and Mingo counties, and maps. Wheeling, West Virginia Geological Survey, 1914.

[This publication is described in the following extract from the printed circular of the Geological Survey: Detailed Report on Logan and Mingo counties, issued under date of Jan. 15, 1915, containing 776 pages + xxi of introductory matter, and illustrated with 15 half-tone plates and 23 figures or zinc etchings in the text; also a case of two maps covering the topography and geology of the entire area of both counties in one sheet. The soil map and Report soon to be published will be sent gratis later to all who receive this volume. In addition to the detailed description and revision of all the rich coal beds and other geologic formations exposed in these counties, the geologic map gives the structure contours and outcrops of the celebrated No. 2 Gas Coal, as also that of several other valuable coal beds, along with many new sections, analyses, etc. Price, with case of maps, delivery charges paid by the Survey, \$2; special combination price with other publications. Extra copies of geologic map, \$1 each, and of the topographic map, 50 cents each.—West Virginia Geological Survey, P. O. Box 848, Morgantown, W. Va.]

Assaying, Chemistry, and Physics

METHODS IN METALLURGICAL ANALYSIS. By Charles H. White. New York, D. Van Nostrand Co., 1915. Price \$2.50. (Gift of Publisher.)

[NOTE.—This volume was prepared primarily for the use of students in mining and metallurgical courses as a text book on the subject indicated by the title, and the treatment is sufficiently comprehensive to serve this purpose admirably; by reason of the careful selection of methods and the excellent arrangement of the material, it should prove equally useful as a handbook for chemists and metallurgists in laboratories of mining and metallurgical plants. In the first chapter is given a general description of laboratory equipment, sampling methods, and the operations of gravimetric and volumetric analysis. Then follow in detail the methods of analysis in the metallurgy of iron and steel, copper, lead and zinc, gold and silver (including the testing of cyanide solutions); the procedures in the analysis of platinum metals, fluxes, fuels (including coal, petroleum, and gas), clay, and some of the minor and rare metals; examination of boiler water; a chapter giving brief instructions for the detection or qualitative determination of the metals; a tabulated statement of the atomic weights, melting points, boiling points, and densities of the elements is also included.—B. A. R.]

CHEMISTRY AND ITS BORDERLAND. By A. W. Stewart. London-New York, 1914.

CHEMISTRY OF PIGMENTS. By E. J. Parry and J. H. Coste. London, 1902.

HANDBUCH DER MINERALCHEMIE. Bd. III, Pt. 5. Dresden, 1914.

Non-metallic Minerals

CANADA. REPORT ON THE NON-METALLIC MINERALS USED IN THE CANADIAN MANUFACTURING INDUSTRIES. Ottawa, Mines Department, 1914.

CANADA. PEAT, LIGNITE, AND COAL. Ottawa, 1914.

General

MINING WORLD INDEX OF CURRENT LITERATURE. Vol. VI, July-December, 1914.

By George E. Sisley, Chicago.

[NOTE.—This volume, like its predecessors, is a classified index of the articles on mining, metallurgy, geology, and allied subjects, which appeared during the last half

of 1914 in periodicals, proceedings of technical and engineering societies, and books published in America, Europe, Africa, and Australia. The articles are classified by subjects, with abundant cross indexing, enabling one to obtain readily a list of current literature on any of the subjects connected with the mining and metallurgical industry. In this volume for the first time are given brief digests of many of the more important articles which have appeared, thereby adding to the usefulness of the publication.—B. A. R.]

MANUFACTURE AND COMPARATIVE MERITS OF WHITE LEAD AND ZINC WHITE PAINTS.

By G. Petit, translated from the French by Donald Grant. London, 1907.

UTILIZATION OF WASTE PRODUCTS. Ed. 2. By Theodor Koller. London, 1915.

EFFICIENT COST KEEPING. Ed. 3. By E. St. Elmo Lewis. Detroit, Burroughs Adding Machine Co., 1914. (Gift of Publisher.)

[NOTE.—One of a type of books of real value issued to advertise the uses of the machine made by this company, but only as an incident in a well-written and useful treatise on cost keeping. The work is a distinct contribution to the literature of efficiency in management.—W. P. C.]

MEN OF SCIENCE AND INDUSTRY. A guide to the biographies of scientists, engineers, inventors and physicians. Carnegie Library of Pittsburgh, Pittsburgh, 1915. (Gift of Carnegie Library of Pittsburgh.)

HAND FIRING SOFT COAL UNDER POWER PLANT BOILERS. Technical Paper No. 80, Bureau of Mines. Washington, 1915.

Company Reports

PHELPS, DODGE & Co. Annual Report, 1914. New York, 1914. (Gift of Company.)

DE BEERS CONSOLIDATED MINES, LTD. Annual Report, 26th, 1914.

BROKEN HILL SOUTH SILVER MINING Co. Reports and Statements of Accounts, for half-year ended Dec. 31, 1914. (Gift of W. E. Wainwright.)

BROKEN HILL PROPRIETARY Co. Reports and Statements of Account, for half-year ending Nov. 30, 1914. (Gift of Company.)

Trade Catalogues

CHICAGO PNEUMATIC TOOL Co., Chicago-New York.

Bull. E-35, Universal electric drills operating on direct or alternating current. Mar. 1, 1915.

Form 212. No. 11 Boyer Riveting Hammer.

KEARNEY & TRECKER Co., Milwaukee, Wis. Catalogue No. 19, Milwaukee Milling Machines.

NEW JERSEY METER Co., Plainfield, N. J. A5, Air meter Bull, 1915.

SUPPLÉE-BIDDLE HARDWARE Co., Philadelphia, Pa. Monel metal. March, 1915.

Notes on Trade Literature

The Jeffrey Manufacturing Company has recently issued Bulletin No. 129-A, illustrating and describing the "Arcwall" coal cutters for the "overcutting" system of mining. It is asserted that the overcutting system is an advanced step in the mining of coal, and has accomplished remarkable results. This bulletin contains valuable information about the design and construction of the "Arcwall" type of mining machine, which is built to cut anywhere above the top of rail, and is especially adapted for cutting out dirt binders that exist in the coal seam. It also contains interesting views of some of the latest installations. Copies may be obtained by writing to the Jeffrey Mfg. Co., 902 North Fourth Street, Columbus, O.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Mar. 10 to Apr. 10, 1915:

Members

- ADAMS, LOUIS WINFIELD, Supt., Rolling Mills, Lehigh Plant,
Bethlehem Steel Co., So. Bethlehem, Pa.
- ALLEN, ARTHUR WATTS, Mine Mgr. Rodeo, San Juan, Argentina,
- BALL, EDWIN MARCOTTEL, Mine Supt., Ishkooda Mines,
Care Tenn. Coal, Iron & R. R. Co., R.F.D. 1, Birmingham, Ala.
- BOLN, CHARLES A., Oil salesman. 135 S. 2d St., Philadelphia, Pa.
- BOYD, DAVID F., Naval Officer. Navy Department, Washington, D. C.
- CAPEERTON, GEORGE HENRY. Charleston, W. Va.
- COGGESHALL, GEORGE W., Chem. Engr., Institute of Industrial Research,
Washington, D. C.
- COLL, CHARLES JOSEPH, Genl. Mgr., Cape Breton Coal, Iron & Ry. Co., Ltd.,
Broughton, Nova Scotia.
- DAVIS, ROY HERNY, Mech. and Elec. Engr., U. S. Coal & Coke Co., Gary, W. Va.
- EVANS, W. A., Genl. Supt. Rock Island Coal Mining Co., Hartshorne, Okla.
- GILL, PHILIP L., Met. Engr. Woodmere, Long Island, N. Y.
- GRANT, LESTER EAMES, Min. Engr. Braden Copper Co., Rancagua, Chile.
- GRISWOLD, W. T., Geol. Philadelphia Gas Co., Pittsburgh, Pa.
- HAMILTON, WILLIAM J., Chief Chem., Granby Cons. Min., Smelt. & Power Co., Ltd.,
Anyox, B. C., Canada.
- HAMLIN, CLARENCE C., Lawyer, 301 Mining Exchange, Colorado Springs, Colo.
- JONES, GEORGE MILTON, Coal mining. Oak Hill, Fayette Co., W. Va.
- LAUER, HERBERT H., Engr. Midvale Steel Co., Philadelphia, Pa.
- LITTLE, ARTHUR DEHON, Chem. and Engr. 93 Broad St., Boston, Mass.
- PALMER, WILLIAM FREDERICK, Miner. P. O. Box 118, Pioche, Nev.
- PAYNE, JAMES HARVEY, Engr. 4004 Dalrymple Ave., Baltimore, Md.
- PETERSEN, OLUF G., Cons. Engr. Somerset, Ky.
- PROUTY, ROSWELL WOODWORTH, Geol., Detroit Copper Mining Co., Morenci, Ariz.
- READ, BILL, Min. Engr. 640 Phelan Bldg., San Francisco, Cal.
- SAYRE, EDWARD A., Genl. Mgr., Eagle Coal Co. Des Moines, Iowa.
- SELMI, LUCIANO, Chief Chem., Otis Steel Co. Cleveland, Ohio.
- SCHIRMER, CHARLES G., Vice-Pres., Argo Reduction & Ore Purchasing Co.,
Boston, Mass.
- SHAW, QUINCY A., Pres., Calumet & Hecla Min. Co.,
12 Ashburton Place, Boston, Mass.
- STROUD, B. K., Petroleum Engr. 250 Grand Ave., Oakland, Cal.
- TOCH, MAXIMILIAN, Paint, Varnish and Chemical Mfr.,
320 Fifth Ave., New York, N. Y.
- TOWNSEND, ROBERT HINCKLEY, Min. Engr., San Luis Mining Co.,
Tayoltita, Dur., Mexico.
- VAN VALKENBURGH, ROBERT RAY, Shift Boss. Perseverance Mine, Juneau, Alaska.
- VENANCOURT, CORNETTE DE, Cons. Engr. 46 Rue de Londres, Paris, France.
- WETMORE, WADE LYNDON, Engr., Granby Cons. Min., Smelt. & Power Co., Ltd.,
Anyox, B. C., Canada.
- WILLIAMS, HENRY GORDON, Coal Mining, Genl. Mgr., Utah Fuel Co.,
Salt Lake City, Utah.
- WILLIAMS, JOHN CHARLES, Chem. Stag Cañon Fuel Co., Dawson, N. M.
- YOSHIWARA, SHIGETAKE, Engr., Daikoku Copper Mine, Azabu, Tokyo, Japan.

Associate Member

- WALSH, THOMAS J., Lawyer. U. S. Senate, Washington, D. C.

Junior Members

- ALLEN, ARTHUR POTTER. 233 College Ave., Houghton, Mich.
- DAVIDSON, HAROLD O., Student. Univ. of Wisconsin, Madison, Wis.

EYOUNG, DJEVAD, Student, Livingston Hall, Columbia University, New York, N. Y.
 FRASER, DONALD DICKINSON, Student..... 233 College Ave., Houghton, Mich.
 GRANT, ULYSSES S., IV., Student..... Harvard University, Cambridge, Mass.
 HANLON, JOHN EDWARD, Student..... 10 Robert St., Toronto, Ont., Canada.
 JOHNSON, FREDERICK ARTHUR, Student,..... 135 Breeding Ave., Ben Avon, Pa.
 KING, ROWLAND BRADBURY, Student..... Theta Tau House, Houghton, Mich.
 KUNDSEN, BJARNE, Student, Univ. of Wisconsin, 644 Frames St., Madison, Wis.
 WOODBRIDGE, ROGER M., Student..... 710 Langdon St., Madison, Wis.
 ZIMMERMAN, STANLEY HUGHSON, Student..... 233 College Ave., Houghton, Mich.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Mar. 10 to Apr. 10, 1915, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Fred Thomas Agthe, Hannibal, Mo.

Proposed by H. Eckfeldt, Henry S. Drinker, Joseph W. Richards.

Born 1886, Hokendauqua, Pa. 1904, Grad., Catasauqua High School. 1905, Grad., Bethlehem Preparatory School. 1909, Lehigh Univ.; E. M. 1909-11, Supt. of Mines, Carolina Barytes Co., Stackhouse, N. C. 1911-13, Engr., Oliver Iron Min. Co., Hibbing, Minn.

Present position: 1913 to date, Engr. in charge of mines and quarries, Atlas Portland Cement Co.

Maurice Merton Albertson, Cobalt, Ont., Canada.

Proposed by H. A. Buehler, C. R. Forbes, G. H. Cox.

Born 1887, Getmore, Kan. 1911, Mo. School of Min. and Met.; B. S. in Min. Engr. 1913-14, Grad. Student, Univ. of Chicago. 1909, Millman's Asst., Coleman Bros. Mine, Aurora, Mo. 1910, Mo. Bureau of Geol. and Mines, Rolla, Mo. 1911-13, Geol., Mo. Bureau of Geol. and Mines. 1914, Mine and mill sampling, Superior Silver Min. Co., Cobalt, Ont., Canada.

Present position: Mine surveyor, Mining Corporation of Canada.

John Barneson, Los Angeles, Cal.

Proposed by Anthony F. Lucas, Max W. Ball., David White.

Born 1862, Caithness, Scotland. 1871-76, Common school. 1876-90, Seaman and shipmaster, through all grades to command. 1890-98, Shipping and commission merchant and shipowner. 1898-1900, Supt., U. S. Army Transport Service.

Present position: 1900 to date, Pres., General Petroleum Co., a number of smaller companies, and the General Pipe Line Co. of California.

Sven Vilhelm Bergh, Chrome, N. J.

Proposed by Sidney Rolle, Robert M. Draper, H. A. Megraw.

Born 1885, Stockholm, Sweden. 1898-1906, Matriculation examination (Real-laroverket, Stockholm, Sweden). 1911, Grad., Royal Technical Univ., Stockholm; Min. Engr. 1906-08, Metallurgical studies, North Easter Steel Wks., Ltd., Middlesbrough, England. 1908-10, Coal mining studies, Bolckow Vaughan, O'Ackland Park Colliery, Bishop O'Ackland, Durham, England. 1911-12, Jernkontorets scholarship: Practical investigation of rock drilling at the Atlas Works, Stockholm, Sweden, and at the Grangesbergs Mines, Sweden. 1911-12, Jernkontorets fellowship: Hammer-drill investigations at the Malmbergets Mines, Lapland, Sweden. (Publications—*Jernkontorets Annaler*.) Draftsman, U. S. Smelting & Refining Co., Chrome, N. J.

Present position: Chief Chemist, Chrome Steel Works.

Ross Herman Boas, Bingham Canyon, Utah.

Proposed by E. P. Jennings, W. G. Swart, W. A. Barnes.

Born 1891, Harrisburg, Pa. 1906-08, Mercersburg Academy. 1908-12, Lafayette College; E. M. 1910, summer, Electrician, D. L. & W. Coal Co., Scranton, Pa., 1911, summer, Engineering Dept., West End Coal Co., Shickshinny, Pa. 1912-13, Chief Engr., Compania Minera La Luz y Los Angeles, Bluefields, Nicaragua. 1913-14, Chief Engr., West End Coal Co.

Present position: 1914 to date, Mine Engr., Utah Metal & Tunnel Co.

Aushelm Cyrus Borgeson, Chisholm, Minn.

Proposed by H. C. Lawrence, A. C. Oberg, William R. Appleby.

Born 1887, St. Peter, Minn. 1905, Minn. North Side High School. 1906, Minn. Business College. 1911, Minn. School of Mines, Univ. of Minn. 1908, summer, Engr. Dept., U. S. Steel Corporation, Hibbing, Minn. 1909, summer, Federal Mining & Smelting Co., Kellogg, Idaho. 1910, summer, Liberty Bell Gold Min. Co., Telluride, Colo. 1911, Shenango Furnace Co., Chisholm, Minn. 1912-14, Resident Engr., Webb Mine, Hibbing, Minn.

Present position: Mining Engineer.

Ralph D. Brown, Harrisburg, Ill.

Proposed by E. C. Lee, C. R. Forbes, Philip N. Moore, G. H. Cox, H. Copeland.

Born 1883, Oxford, Ohio. 1900-04, Miami Univ., Oxford, O.; A. B. 1907-09, Special student in C. E., Ohio State Univ. 1904-06, Prof. Science, N. E. Ohio Normal College, Canfield, O. 1906-07, Asst. Engr., B. & O. R. R. and Coal and Coke Ry. 1909-11, Min. Engr., O'Gara Coal Co., Harrisburg, Ill. 1911-12, Instructor, School of Mines, Rolla, Mo. 1912-14, Chief Engr., O'Gara Coal Co.

Present position: 1914 to date, Asst. Genl. Supt., Trustees, O'Gara Coal Co.

Frank Cameron, Salt Lake City, Utah.

Proposed by Wallace G. Woolf, Dorsey A. Lyon, Robert S. Lewis.

Born 1889, Salt Lake City, Utah. 1909, Salt Lake High School, Scientific Course. 1913, Univ. of Utah; B. S. Summers of 1908-12, and year 1913-14, Surveying, millman, transitman, mining, Nev. Cons. Copper Co.

Present position: 1914 to date, Fellow Metallurgical Research, Univ. of Utah; Master's Degree.

Donald George Campbell, Seattle, Wash.

Proposed by Joseph Daniels, Milnor Roberts, J. L. McAllen.

Born 1887, Port Arthur, Canada. 1908-12, Whitman College; B. S. in Chem. 1912-13, Columbia Univ.; M. A.; 1913-14, E. M. 1902-03, Mill work, Big Master Mining Co., Kenora, Canada. 1903-07, Machinist and draftsman, Hunt Co., Walla Walla, Wash. 1907-08, Electrician and millman, Chase Bagdad Co., Atlanta, Idaho. 1908-12, Mining and milling, Atlanta, Idaho. 1914, Asst. Engr., N. Y. Board of Water Supply, N. Y.

Present position: 1914 to date, Instructor in Metallurgy, Univ. of Wash.

Douglas Clark, Stewart, Idaho.

Proposed by J. C. Branner, C. F. Tolman, Jr., D. M. Folsom.

Born 1893, Sacramento, Cal. 1914, Stanford Univ., Geol. and Min.; A. B.

Present position: 1914 to date, Flotation work, Ontario Mining Co.

Ronald Clark, New York, N. Y.

Proposed by H. W. Hardinge, R. B. T. Kiliani, H. W. Spicer.

Born 1892, Chicago, Ill. 1908, Armour Scientific Academy. 1912, Armour Institute of Technology; B. S.

Present position: 1912 to date, Hardinge Conical Mill Co.

Joseph D. Davis, Washington, D. C.

Proposed by Van H. Manning, Albert H. Fay, W. A. Williams.

Born 1882, Middleport, Ohio. 1897-1901, Public schools, Columbus, Ohio. 1901-05, Ohio State Univ. Completed work required for degree of B. A., and in addition work offered in metallurgy and mine engineering, under Professors Lord and Ray of Ohio State Univ. 1906, Chemist, Columbus Iron & Steel Co., Columbus, O. 1906-07, Chemist, Love Bros. Paint Co., Dayton, O. 1907-11, Jr. Chemist, U. S. Geological Survey.

Present position: 1911 to date, Chemist, Bureau of Mines.

Hugh Dolan, Pottsville, Pa.

Proposed by E. C. Luther, A. W. Sheaffer, William A. Cochran.

Born 1874, Pottsville, Pa. 1888, Local schools. 1888-91, Mt. St. Mary's College, Emmitsburg, Md. 1892-96, A. B. Cochran & Son.

Present position: 1896 to date, engaged in mine development, shaft sinking and tunneling for various anthracite mining companies, principally the Philadelphia & Reading Coal & Iron Co.

Frank E. Downing, Chisholm, Minn.

Proposed by H. C. Lawrence, Frank G. Jewett, A. P. Silliman, L. R. Salsich, E. J. Longyear.

Born 1878, St. Charles, Minn. 1892-95, St. Charles, Minn., High School. 1897-1901, 1903-04, College of Engrg., Univ. of Minn.; C. E. 1899-1900, St. Paul office, U. S. Engineers, recorder on stream gauging, Otter Tail Lake, Minn. Recorder and transitman on flowing surveys, same office. 1901-03, Surveyor with E. J. Longyear, Exploration Co., Hibbing, Minn. 1904, with G. L. Wilson, C. E. for Twin City Rapid Transit Co., Minneapolis. 1904-05, Engr. with E. J. Longyear. 1905-06, Engr., Meriden Iron Co., Hibbing, Minn. 1906-11, Engr., Oliver Iron Min. Co., Hibbing, Minn.

Present position: 1911 to date, Asst. to Genl. Mgr., Shenango Furnace Co.

Oliver Camille Dupuis, Yellow Jacket, Idaho.

Proposed by John B. Hastings, Ralph Nichols, Joseph J. Taylor.

Born 1882, Canada. 1890-96, Public school, Butte, Mont. 1896-98, College of St. Paire, Canada. 1903-07, International Correspondence School. 1909, Washington State College. 1900-05, Miner in Montana, Idaho, and Washington. 1905-06, Oregon & Idaho Mining Co. 1907, Rabbit Foot Mining & Milling Co. 1910, Golden Age Mining Co. 1910-14, Musgrove Mining Co.

Present position: Foreman, Mandarin Mines Corporation.

George W. Gehres, Republic, Pa.

Proposed by George H. Morse, John E. Perry, H. P. Zeller.

Born 1879, Llewellyn, Pa. 1897, Grad., Normal School, Kutztown, Pa. Course with I. C. S., Scranton, Pa.; Min. Engrg. 1897-1900, Engrg. Departments, Midvalley Coal Co., Wilburton, Pa.; Philadelphia & Reading Coal & Iron Co., Ashland, Pa. 1900-05, Supt. and Engr., Bondo Coal Co., Mt. Carmel, Pa. 1905-06, Practical miner and mine foreman, various companies in Pennsylvania. 1906-08, Supt., Old Colony Coal & Coke Co., Pittsburgh, Pa. 1908-09, Genl. Supt., Old Colony Coal & Coke Co., Pittsburgh, Pa.; Glen Eastern Coal & Coke Co., Moundsville, W. Va. 1909-13, Supt., Columbia Coal & Coke Co., Ligonier, Pa. 1913-14, Supt., Greensburg-Connellsville Coal & Coke Co., Ligonier, Pa.

Present position: Genl. Supt., Northern Coal Mines, Republic Iron & Steel Co.

Herbert William Gepp, Broken Hill, Australia.

Proposed by George C. Stone, J. H. Troutman, J. A. Van Mater.

Born 1877, Adelaide, So. Australia. 1890-93, Prince Alfred College. 1894-97, Melbourne University Chemical School. 1894-95, Chemist, finally Mgr., Explosives and Superphosphate Works near Melbourne (branch of Nobel's Explosives Co. of Glasgow, Scotland). 1898, Ardeer Works of Nobel's Explosives Works, Scotland. 1906-07, Acid Plant Supt., Zinc Corporation, Broken Hill, including design and construction of chamber acid plant.

Present position: 1907 to date, Genl. Mgr., Amalgamated Zinc, Ltd.

Ira L. Greninger, Miami, Ariz.

Proposed by Herman C. Hase, J. K. Foreman, H. Kenyon Burch.

Born 1878, Moscow, Idaho. Common schools of Oregon. 1905-06, Asst. Supt., Greenback Mine, Greenback Gold Min. Co., Josephine Co., Ore. 1907, Shift boss, Blue Lodge Copper Min. Co., Hutton, Cal. 1908-11, Genl. Foreman, underground work, Balaklala Mine, for Balaklala Cons. Copper Co., Kimberley, Cal.

Present position: 1911 to date, Minerals Separation American Syndicate, Ltd.

LeRoy M. Gross, New York, N. Y.

Proposed by Harry J. Wolf, Kirby Thomas, F. W. Traphagen.

Born 1890, New York City. 1909-13, Columbia University. 1913-14, Colo. School of Mines; E. M.

Present position: With Theodore Gross since graduation and spent previous summers with this office; work has consisted of mine reports, the checking and figuring of costs, assay plans, power installation, operating contracts, etc.

George Townsend Harley, Morenci, Ariz.

Proposed by Arthur Crowfoot, T. B. Counselman, Frank A. Ayer.

Born 1889, New York, N. Y. 1909-12, Univ. of Colo.; specialized in chemistry, geology and economics. 1912, Registered in Univ. of Wis., in general technical work, extension division. 1912-14, Millman, chemist, engineer and genl. supt., in charge mill experimental work, prospect churn drilling, mining and office work, Tularosa Copper Co., Tularosa, N. M. 1914, Assayer, Cripple Creek, Colo. 1914-15, Millman, Aurora Cons. Mines Co., Aurora, Nev.; solutions and tube mills.

Present position: Oil flotation, Arizona Copper Co.

Daniel Harrington, Pittsburgh, Pa.

Proposed by J. J. Rutledge, L. K. Smith, Van H. Manning.

Born 1878, Denver, Colo. 1896, Grad., Denver High School. 1900, Grad., Colorado School of Mines; E. M. 1900-07, Mine Surveyor, Construction Supt., Chief Engr., Utah Fuel Co. in Utah coal fields. 1907, Construction Supt., United States Smelt., Ref. & Min. Co., Eureka, Utah. 1907-10, Cons. Engr., Salt Lake City, Utah. 1910-14, Genl. Supt., Big Horn Collieries Co., Crosby, Wyo.

Present position: 1914 to date, Min. Engr., U. S. Bureau of Mines.

George Henry Ronald Harris, London, Ont., Canada.

Proposed by J. L. McAllen, Milnor Roberts, Joseph Daniels.

Born 1873, London, Ontario. 1890-04, Grad., Royal Military College, Kingston, Canada. 1894-97, Post-graduate work, Michigan College of Mines. 1897-1900, Min. Engr., Greenwood, B. C. 1900-1901, Min. Engr., Harrison & Barchard, Mossamides, Angola, W. Africa. 1901-03, Min. Engr., East Africa Syndicate. 1903-04, In charge exploration work, Cassinga Concessions. 1905-08, Mgr., Northland Mine, Rib Lake, Ont. 1908-10, Exploration work, L. C. Thomson, Bolivia.

Present position: 1913 to date, Mgr., Gold Bullion Mine, Knik, Alaska.

George Robert Hilby, Salinas, Cal.

Proposed by J. C. Branner, D. M. Folsom, James C. Ray.

Born 1890, Monterey, Cal. 1914, Stanford Univ.; A. B. in Geology and Mining. 1911-12, H. D. Severance, C. E., Monterey. 1913, C. H. Sanders, Los Angeles. 1914, H. D. Severance, Monterey.

Present position: 1914 to date, draftsman and transitman, Lou G. Hare.

Harold Howe Hodgkinson, Franklin Furnace, N. J.

Proposed by R. M. Catlin, J. F. Kemp, Charles P. Berkey.

Born 1886, Toronto, Canada. 1901-05, Cleveland Central High School. 1906-07, Mass. Inst. of Tech. 1907-10, Columbia Univ., School of Mines; E. M. 1905, Standard Tool Co., Cleveland, O. 1906-07, Lake Shore & Michigan Southern R. R. 1909, Copper Range Cons. Copper Co.

Present position: 1910 to date, Min. Engr., New Jersey Zinc Co.

Francis Augustus Holiday, Whitwell, Herts, England.

Proposed by David T. Day, Alfred H. Brooks, George Otis Smith.

Born 1873, London, England. 1885, Westminster School, London. 1893, Royal College of Science and School of Mines, London. 1896-99, Demonstrator. Associate Royal College Science, London. Member Geology Society, London. Member Institute Petroleum Technologists, London.

Present position: 1909 to date, with Sir Boverton Redwood, Bart., of London, making geological and technical examinations in Venezuela, Trinidad, Mexico, Texas, and various Russian and Siberian fields.

Chin-Tao Huang, Hanyang, China.

Proposed by Y. C. Kuang, Thomas T. Read, George I. Adams.

Born 1889, Amoy, China. 1899-1906, Amoy Anglo Chinese College, China (grad.). 1906-08, College, prepared to enter the professional department, Tientsin, China. 1908-12, Min. and Met. department, Pei-yang Univ., Tientsin. 1912-14, School of Mines, Columbia Univ., New York; M. A. 1909, Geological work, Western Hill, Pekin, China. 1910, Metallurgical works, Hanyang Iron & Steel Wks., China. 1912, Surveying work, Tongshan Colliery, China.

Present position: 1914 to date, Asst. Engr., Hanyang Iron & Steel Works.

Gustave A. Jahn, Brooklyn, N. Y.

Proposed by J. F. Kemp, Charles P. Berkey, Arthur L. Walker.

Born 1886, Brooklyn, N. Y. 1905-07, Cornell Univ. 1907-11, Columbia University; E. M. 1911-13, Miami Copper Co. 1914, Breitung & Co., Ltd.

Present position: Disengaged.

Arthur William Koch, Mascot, Tenn.

Proposed by J. N. Houser, Robert Ammon, H. K. Sherry.

Born 1887, Denver, Colo. 1904-05, High school, Denver, Colo. 1906-11, Colorado Iron Works Co., Denver, Colo. 1911-12, Kennecott Mines Co., Kennecott, Alaska. 1912-13, Utah Copper Co., Magna Plant. Butte & Superior Copper Co., Butte, Mont. 1913-14, La Fe Mining Co., Ltd., Guadalupe, Zacatecas, Mexico.

Present position: American Zinc Co.

James A. Lane, New York, N. Y.

Proposed by Howard F. Chappell, Charles H. MacDowell, George S. Rice.

Born 1890, Leavenworth, Kan. 1908-13, Specialized in Geology at Univ. of Chicago; S. B. 1908-09, Asst. to Prof. A. C. Trowbridge in his geological work in the Owens River Valley, Cal.

Present position: 1912 to date, Mineral Products Co.

Carl Nelson Nelson, San Juancito, Honduras, C. A.

Proposed by Robert Peele, Adam R. Gordon, J. F. Kemp.

Born 1883, Paris, France. 1900, Public and high schools. 1901-05, Columbia Univ.; E. M. 1905, Practical work in New York State. 1906, Surveyor and Assayer. Cieneguita Copper Co., Sahauripa, Sonora, Mexico. 1907, Practical work at Bisbee and Cananea. 1908, Mine Supt., Hinds Cons. Min. Co., Santa Barbara, Chih., Mexico. Mine Foreman, Cobalt Central Mine, Cobalt. 1909, Supt., Total Wreck Lease, Pantano, Ariz. 1910, Exploration and prospect work in Sonora, Mexico. 1911, Engr., Sunset Development Co., La Barranca, Sonora, Mexico.

Present position: 1911 to date, Mine Foreman, New York & Honduras Rosario Mining Co.

Howard R. Norsworthy, Kinchasa, Belgian Congo, W. Africa.

Proposed by J. F. Kemp, S. H. Ball, D. M. Riordan.

Born 1882, New York City. 1896, New York public school. 1896-1900, Pulitzer Scholar at Horace Mann High School. 1900-04, Pulitzer Scholar, School of Mines, Columbia Univ.; E. M. 1904-07, Assayer and Surveyor for several companies in Western States. 1907, Engr., with D. M. Riordan. 1908, Miner, Centennial Eureka Mine, Eureka, Utah. 1909, Surveyor, Mountain Copper Co. 1910-12, Surveyor and Engineer, United Comstock Pumping Assn., Virginia City, Nev. 1912-13, Examination work in California and Oregon. 1913-14, Engr., Butters Salvador Mines Co., Salvador, C. A.

Present position: Min. Engr., Société Internationale Forestière et Minière du Congo.

Earl Oliver, Bartlesville, Okla.

Proposed by Max W. Ball, Charles T. Lupton, Roswell H. Johnson.

Born 1878, Butler Co., Pa. 1894, Penn. Common Schools. 1895-98., Slippery Rock Normal, Penn. 1908-10, Transylvania University, Lexington, Ky.; B.B. L. 1890-95, Operated oil wells. 1898-1900, Operated oil wells. 1900-02, Tool driver on drilling wells. 1902-09, Oil-well driller, contractor, lease supt., oil producer. 1910-13, Oil property inspector, Wolverine Oil Co., Tulsa, Okla.

Present position: 1913 to date, oil business.

Alfred Pick, New York, N. Y.

Proposed by Roswell H. Johnson, E. N. Breitung, C. B. Dunster.

Born 1876, Slavonia, Austria. 1896, "Matura," Kgl. Realschule Karlovac-Rakovac, Croatia, Austria. 1900, "Diplom Ingenieur," Koenigl. Technische Hochschule, Hannover, Germany. 1900-01, Berliner Mach. Bau Act. Gesell., Berlin. 1902, L' University de Genève, Switzerland.

Present position: Breitung & Co., Ltd.

Thomas S. Roberts, Lyon Mountain, N. Y.

Proposed by C. E. Addams, W. S. Boyd, L. S. Cates.

Born 1884, Wilkes-Barre, Pa. 1904, Grad., Plymouth Township High Schools, Pa. 1904-07, Min. Engrg. Dept., Delaware & Hudson Co. 1907-09, Min. Engrg. Dept., Delaware, Lackawanna & Western. 1909-14, De Beers Consolidated Mines, Ltd., Kimberley, So. Africa.

Present position: Underground Supt., Chateaugay Ore & Iron Co.

William Paxton Roberts, Mineville, Essex Co., N. Y.

Proposed by Knox Taylor, A. S. Hummell, Victor Angerer.

Born 1892, Philadelphia, Pa. 1904-10, Haverford School, Haverford, Pa. 1910, Princeton Univ.; A. B.

Present position: 1914 to date, Assistant Mine Surveyor, Witherbee, Sherman & Co.

Howard C. Smith, Cripple Creek, Colo.

Proposed by F. K. Brunton, F. M. Taylor, D. W. Brunton.

Born 1889, La Salle, N. Y. 1907, Grad., Cripple Creek High School. 1913, Grad., Colo. School of Mines; E. M. 1913, Assisted in mill test for Vindicator Cons. Gold Min. Co.

Present position: 1914 to date, Solution man, Victor Mill, Portland Gold Mining Co.

William Sutton, London, S. W., England.

Proposed by David T. Day, Alfred H. Brooks, George Otis Smith.

Born 1875, London, England. Technical engineering course, Dulwich College, England. Practical training; pupil with Messrs. James Simpson, Ltd., London, England. Practical experience, in the oil fields of Russia, Roumania, Galicia, Alsace, Italy, Egypt, and various parts of the United States of America. With Sir Boverton Redwood, Bart., in the examination, reporting on, and management of oil properties in various parts of the world. Member of the Institution of Mining and Mechanical Engineers, member of the Institution of Petroleum Technologists.

Present position: Tech., Sir Boverton Redwood.

Robert E. Tally, Jerome, Ariz.

Proposed by Will L. Clark, C. V. Hopkins, W. A. Clark.

Born 1877, Virginia City, Nev. 1899, Grad., Mining School, Nevada State Univ.; B. S. 1899, Assayer and chemist, Virginia City, Nev. 1900, Assayer and chemist, Reno, Nev. 1900, Assayer, chemist and engr., Rossland, B. C., Canada. 1900-06, Miner, shift boss, foreman, engr., Rossland, B. C.; Mace, Idaho; Pine, Idaho; Keswick, Cal.; Sumpter, Ore.; Butte, Mont. 1906-07, Supt., Cal. 1907-08, Asst. Supt., Jerome, Ariz.

Present position: 1908 to date, Mine Supt., United Verde Copper Co.

Frank Tickner, Sharpsville, Pa.

Proposed by W. A. Barrows, Jr., Carl Zapffe, George L. Collard.

Born 1859, Warwickshire, England. General education with chemistry in private schools in England. Seventeen years with Illinois Steel Co., So. Chicago, blast furnace and foundry work, with some research work on physical and chemical properties of iron. Supt. of iron, steel, and brass foundries. Five years Lackawanna Steel Co., Buffalo. Built foundry and established casting of ingot molds from direct metal. Supt. of iron, steel, and bronze foundries. Work on Babbitt metal.

Present position: Genl. Supt., Valley Mould & Iron Co., blast furnace and direct-metal casting of ingot molds.

Cheng-Fu Wang, New York, N. Y.

Proposed by William Campbell, Arthur L. Walker, J. F. Kemp.

Born 1890, Ningpo, China. 1904-07, Preparatory Department, Pei Yang University, Tientsin, China. Degree, Pa-kung Engineering. 1907-10, Mining and Metallurgy in the School of Engineering, Pei Yang University, Tientsin, China. Degree, Chin-Shih. 1910, Conferred Hanlin Fellow of the Hanlin Academy by the Imperial Decree. 1911-12, Metallurgy and mining, in the School of Mines, Columbia Univ.; A. M. 1907, Geology of coal deposits, Western Hills, Peking, China. 1908, Iron and coal mining and in steel works, Han-Yeh-Ping & Co., China. 1909, Geology of iron deposits, and investigating iron works in Pingting-chou, Shansi, China. 1910-11, Committee on Mining and Engineering in the Hanlin Academy, China.

Present position: 1912 to date, graduate student in the School of Mines, Columbia Univ.

Arthur John Weinig, Telluride, Colo.

Proposed by Charles A. Chase, Arthur Winslow, John V. N. Dorr.

Born 1883, Durango, Colo. 1908, Colo. School of Mines; E. M. 1908-10, Sampler, Durango plant, A. S. & R. Co. 1910-11, Shenandoah Dives Leasing Co., Silverton, Colo.

Present position: 1911 to date, Mill Supt., Liberty Bell Gold Mining Co.

Associate Members

Leland Day Albin, New York, N. Y.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1875, Mercer, Pa. High school, academy and business college. Eighteen years' experience in mining and quarrying machinery in Ingersoll-Rand Co., sales department.

William Rupert Elliott, Denver, Colo.

Proposed by W. L. Saunders, Henry Lang, George A. Howells.

Born 1873, Montreal, Quebec, Can. 1879-1887, Senior School, Montreal. 1898-1901, Mech. Drawing and Elec. Engrg., Night Schools of Montreal. 1902-03, Mech. Engrg., Night Schools, Sherbrooke. 1887-99, Central Vermont Ry., Montreal. 1899-1912, Canadian Ingersoll-Rand Co., Montreal and Sherbrooke.

Present position: 1912 to date, Treas., J. George Leyner Engrg. Works Co.

John H. Patterson, Dayton, Ohio.

Proposed by B. B. Thayer, Arthur Williams, Bradley Stoughton.

Born 1844, Dayton, Ohio. Three years Miami College, Oxford, Ohio. 1867, Dartmouth; A. B. 1900, Received the decoration of the Legion of Honor from French government.

Present position: 1885 to date, Pres. of National Cash Register Co.

Junior Members

John J. Croston, Stanford University, Cal.

Proposed by G. H. Clevenger, H. W. Young, J. C. Ray.

Born 1893, Boston, Mass.

Present position: Student, Stanford University.

Arthur Chester Dammann, Denver, Colo.

Proposed by Harry J. Wolf, William R. Chedsey, F. W. Traphagen.

Born 1889, Chicago, Ill. 1907, Crane Tech., Chicago. 1909, Lewis Institute, Chicago. Western Electric Co. Chicago Telephone Co. Consolidated Indiana Coal Co. Western Union Telegraph Co. Independence Gold Mining Co.

Present position: Student, Colorado School of Mines.

Eric Roy Emmerson, Toronto, Ont., Canada.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1889, Port Arthur, Ont., Canada. 1904-08, Port Arthur High School. 1914, summer, Buffalo Mines Co., Cobalt, Ont., Canada, High Grade Mill.

Present position: 1908 to date, student, Univ. of Toronto.

William Teasdale Hall, Toronto, Ont., Canada.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1893, Toronto, Ont., Canada. 1908-11, Harbord Collegiate Inst. 1912, O'Brien Mine, Gowganda, ore sorting. 1913, McIntyre & Mining Co., Porcupine, sampling. 1914, Mines Branch of Geological Survey.

Present position: 1911 to date, student Univ. of Toronto.

Walter Elmer Rohrer, McAlester, Okla.

Proposed by E. S. Dickinson, C. M. Young, George M. Brown.

Born 1889, Parsons, Kan. High School, Parsons, Kan. 1906-07, Student, Missouri School of Mines, Rolla, Mo. 1909-10, Stationary Engr., Busby Hotel, McAlester, Okla. 1912, three months, Mining Engr., Osage Coal & Mining Co., McAlester, Okla. 1913-14, 10 months, practical miner, Victoria Coal & Min. Co., Henryetta, Okla.

Present position: Student, Univ. of Kansas, Lawrence, Kan.

Arent Henry Schuyler, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, G. A. Roush, Henry S. Drinker.

Born 1893, Nutley, N. J. 1901-07, Public school, New York. 1907-11, Townsend Harris, N. Y.; Dwight School, N. Y.; Hamilton School, Brooklyn, N. Y. 1914, Asst. in Knudson Experiment at Canadian Copper Co., Copper Cliff.

Present position: Student, Lehigh University, Met. Engrg.

James Brownlee Stitt, Haileybury, Ont., Canada.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1889, Kemptville, Ont., Canada. 1902-06, Renfrew Collegiate Inst. 1907-09, Victoria Mining Co., Cobalt, Ont., Canada., bookkeeping, etc. 1909-11, in charge of operations, Moose Horn Mines, Ltd. 1912, Mining and milling work, Cobalt Lake Co. 1913, In charge of Moose Horn Mines, Ltd. 1914, Geol. Survey of Canada.

Present position: 1911 to date, Student, Univ. of Toronto, Engrg.

Frederick Sylvester Wright, New York, N. Y.

Proposed by Arthur L. Walker, J. F. Kemp, Charles P. Berkey.

Born 1890, Electric, Mont. 1909-10, Preliminary and location work, Chicago, Milwaukee & Puget Sound Ry.

Present position: Student, Columbia University, School of Mines.

George W. Zacharias, Stanford University, Cal.

Proposed by J. C. Branner, C. F. Tolman, Jr., James C. Ray.

Born 1895, Harper, Kan.

Present position: Student, Stanford Univ.

Change of Status—Junior Member to Member

Allyne Francis Pratt, Portland, Ore.

Born 1888, Portland, Ore. 1894-1905, Public schools of Portland, Ore. 1906, Oregon Agriculture College. 1908-10, Colorado School of Mines. 1913, Montana School of Mines; E. M. 1914, Engr., Jacobson Bade Construction Co.

Present position: Construction work, Alaska-Gastineau Mining Co.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Mar. 10 to Apr. 10, 1915. This list, together with the list published in *Bulletin* No. 100, April, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of Apr. 10, 1915.

ACKER, LOUIS K., JR.	2203 Oliver Bldg., Pittsburgh, Pa.
ALDASORO, ANDRES	6a Balderas No. 92, Mexico City, Mexico.
AMIDON, R. G.	Care Queen of the West Mines Co., Cornucopia, Ore.
ANDERSON, ANDREW P.	1504 Hobart Bldg., San Francisco, Cal.
ANDERSON, P. S.	P. O. Box 446, Baker, Ore.
ANDERSON, ROBERT JOHN	2153 Fairmont Road, S. E., Cleveland, Ohio.
ARMS, CHARLES S.	1801 Adams St., Toledo, Ohio.
BAIN, H. FOSTER	734 Salisbury House, London, E. C., England.
BARBER, G. M.	42 Clapham Road, Bedford, England.
BARRETT, THEODORE H.	Wanakah Mining Co., Ouray, Colo.
BATCHELLOR, STILLMAN	Globe Cons. Mining Co., Dedrick, Trinity Co., Cal.
BECKWITH, H. T., Geol.	405 Kipling St., Palo Alto, Cal.
BELL, W. L.	130 Kennedy St., Bradford, Pa.
BLACK, THOMAS B.	2101 Jackson St., Sioux City, Iowa.
BLAKEMORE, G. H.	4 Bridge St., Sydney, N. S. W., Australia.
BODFISH, FREDERIC V.	P. O. Box 1357, Salt Lake City, Utah.
BOOTH, EVERETT L.	349 W. 145th St., New York, N. Y.
BOTTOMLEY, HERBERT	Boksburg, South Africa.
BOYLE, C., JR.	522 Connell Bldg., Scranton, Pa.
BRADLEY, D. H., JR.	P. O. Box 93, Prescott, Ariz.
BRALY, NORMAN B.	P. O. Box 573, Butte, Mont.
BRAYTON, COREY C., Min. Engr. and Genl. Mgr., Shovel Creek Gold Dredging Co.,	
	433 California St., San Francisco, Cal.
BREINL, JOSEPH C.	840 South Hope St., Los Angeles, Cal.
BRINSMADE, ROBERT B., la Pensador Mexicano 1, Pueblo, Pue., Mexico, via Vera Cruz.	
BROWN, JAMES F., 2D.	Box 604, Goldfield, Nev.
BUNTING, DOUGLAS	93 W. Union Street, Wilkes-Barre, Pa.
BURBRIDGE, T. B.	German American Trust Bldg., Denver, Colo.
CARNAHAN, G. H., Vice-Prest., Intercontinental Rubber Co. of New York,	
	Room 1134, 17 Battery Pl., New York, N. Y.
CARNAHAN, J. S.	Cadiz, Ohio.
COFFEY, GEORGE T.	Care Yukon Gold Co., Dawson City, Y. T., Canada.
COLLINS, GLENVILLE A.	1201 L. C. Smith Bldg., Seattle, Wash.
CORWIN, FRANK R.	759 12th St., Bisbee, Ariz.

- CRAMPTON, THEODORE H. M. Box 783, Santa Monica, Cal.
 CROSS, WILBUR L., JR., Chief Engr., Buck Run Coal Co., Minersville, Pa.
 CUNNINGHAM, WALTER H., Cons. Engr. Huntington, W. Va.
 DAGGETT, ELLSWORTH. 444 E. South Temple St., Salt Lake City, Utah.
 DAVIS, JOHN A. Naturita, Colo.
 DOBSON, CHRISTOPHER G. 444 W. Broadway, Butte, Mont.
 DOUGLAS, THEODORE. 245 W. 55th St., New York, N. Y.
 DOWE, JOHN H., Champion (Nigeria) Tin Fields, Ltd., Care Postmaster,
 Minna, Northern Nigeria.
 DOWNS, FLETCHER G. Chatham, Morris County, N. J.
 EDMONSON, H. W., Supt. Concheno Mining Co., Temosachic, Chih., Mexico.
 EMERY, A. B., Messina Transvaal Development Co., Ltd.,
 Messina, Transvaal, So. Africa.
 EMMONS, N. H. 2 Rector St., New York, N. Y.
 EVERED, N. J. North Thompson Mine, Timmins, Ont., Canada.
 FARISH, F. G. 315 Colorado Bldg., Denver, Colo.
 FERGUSON, CLAUDE. Chaparral, Ariz.
 FITCH, MAX B. 177 S. E. Boulevard, Corona, Cal.
 FOUST, THOMAS B. Mgr., La Follette Iron Co., La Follette, Tenn.
 FOX, WALTER V. Ray Consolidated Copper Co., Ray, Ariz.
 FROLLI, ALBERT W., Engr. and Accountant, Presidio Mining Co., Shafter, Texas.
 GAY, WARE B. Mayo Land & Bldg. Co., Richmond, Va.
 GEMMELL, R. C. P. O. Box 1775, Salt Lake City, Utah.
 GIBBS, GEORGE H. Stynechale Ave., Earlsdan, Coventry, England.
 GOODRICH, ROBERT R. 421 W. 121st St., New York, N. Y.
 GRAUPNER, M. F. Bunker Hill and Sullivan Mine, Kellogg, Idaho.
 GROSS, JOHN. 541 Corona Street, Denver, Colo.
 HALL, BENJAMIN MORTIMER. 521 Peters Building, Atlanta, Ga.
 HAMILTON, H. T. Moctezuma Copper Co., Nacozari, Son., Mexico.
 HASE, HERMAN C. Instructed to hold all mail.
 HAY, HENRY. Chuquicamata, via Antofagasta, Chile.
 HELLER, M. W. 847 E. Colfax Ave., Denver, Colo.
 HEWITT, E. A. 1122 California Ave., Butte, Mont.
 HIESTER, ARTHUR J. 1340 Garfield St., Denver, Colo.
 HILLS, VICTOR G. 415 McPhee Bldg., Denver, Colo.
 HOCKING, RICHARD O. Box 1056, Miami, Ariz.
 HOLLIS, HENRY L. 1025-122 S. Michigan Ave., Chicago, Ill.
 HOTCHKIN, M. W., Care Tough Oakes Gold Mines, Ltd., Kirkland Lake, Ont., Canada.
 HUGHES, WILSON W. 534 I. W. Hellman Bldg., Los Angeles, Cal.
 HYNES, DIBRELL P. 1025 Peoples Gas Bldg., Chicago, Ill.
 JEFFS, LEWIS A. 404 Atlas Blk., Salt Lake City, Utah.
 JONES, EDWARD B. Lehi, Utah.
 KANE, JOHN I. 711 Mills Building, El Paso, Texas.
 KLOPSTOCK, PAUL. Care Elks Club, 108 W. 43d St., New York, N. Y.
 LADD, DAVID H., Care Elder, Smith & Co., Ltd., 73 Basinghall St., London, E. C.,
 England.
 LAIRD, GEORGE A., 10 Queen's Chambers, 4 Dalley St., Sydney, N. S. W., Australia.
 LANAGAN, WILLIAM H. Monadnock Bldg., San Francisco, Cal.
 LAWSHE, VERNER T. Apartado 63, Chihuahua, Mexico.
 LINDEMUTH, LEWIS B. Natural Bridge, N. Y.
 LINDSLEY, H. 60 Broadway, New York, N. Y.
 LOW, V. F. STANLEY. Care The National Bank, Perth, West Australia.
 MANN, WILLIAM SEWARD, Care Shan Tsz Mining Co., El Dorado, El Dorado Co., Cal.
 MANNING, VAN H. 3556 Macomb St., Washington, D. C.
 MASTERS, JOHN E. 627 N. 4th St., Tucson, Ariz.
 MAYER, LUCIUS W. 14 Wall St., New York, N. Y.
 MAZANY, M. S. International Smelting Co., Miami, Ariz.
 MACDONALD, WILLIAM T. Care Ray Cons. Copper Co., Hayden, Ariz.
 MACFARLANE, RIENZI W. P. O. Box 102, Parral, Chih., Mexico.
 MERRIMAN, T. C. 198 Winthrop Ave., New Haven, Conn.
 MOORE, CARL F. P. O. Box 1785, Salt Lake City, Utah.
 MOORE, L. D. Chatham, N. J.
 MOORE, REDICK R. 427 Judge Bldg., Salt Lake City, Utah.
 MOXHAM, ARTHUR J. Room 1210, 2 Rector St., New York, N. Y.
 MUIR, THOMAS KING. Tonopah, Nev.
 MURDOCK, JOHN A. W., Cerro de Pasco Mining Co., Casilla No. 309, Lima, Peru.

- NAGEL, FRANK J., Care Cia. de Minerales y Metales, S. A., Monterey, N. L., Mexico.
- NICHOL, W., Kroonstad Coal Estate Co., Ltd., Vierfontein, Orange Free State,
So. Africa.
- NICHOLS, J. CLAYTON, Min. Engr., Lessee, Danao Coal Mines,
Danao, Cebu Island, P. I.
- NISHIO, KEIJIRO.....86 St. James Ave., Boston, Mass.
- OUGH, S. W.....Care A. H. Ough, Esk Hall Heights, Yorks., England
- PATCHELL, FREDERICK J.....Dome Mines, Ltd., So. Porcupine, Ont., Canada.
- PERKINS, EDWIN T., Mine Operator.....2105 Moffett St., Joplin, Mo.
- PIGOTT, CURTIS.....Care American Smelting & Refining Co., Murray, Utah.
- PLUMB, EDMUND T.....P. O. Box 400, Terryville, Conn.
- PORTUGAL, JOSEPH H.....30 S. State St., Salt Lake City, Utah.
- POWER, HAROLD T., Room 625, Call Bldg., 74 New Montgomery St.,
San Francisco, Cal.
- PRYOR, JAMES.....Wallaroo Mines, Kadina, So. Australia.
- QUAYLE, THEODORE W.....Tyrone, N. M.
- RAINSFORD, R. S.....Care General Motors Co., Detroit, Mich.
- RAY, FRANK A.....515 Harrison Bldg., Columbus, Ohio.
- RHODES, W. B.....Golden, Colo.
- RICHARDS, J. V.....1024 Old National Bank Bldg., Spokane, Wash.
- RICHARDSON, W. V.....2849 Prince Street, Berkeley, Cal.
- RICKARD, T. A., Editor of *Mining Press*.....420 Market St., San Francisco, Cal.
- RODGERS, M. K.....Brentwood Park, Westgate, Cal.
- RODGERS, GEORGE R., Min. Engr.....47 St. Vincent St., Toronto, Ont., Canada.
- ROLKER, CHARLES M.....126 Riverside Drive, New York, N. Y.
- SCHINDLER, D. F.....215 N. Murray St., Madison, Wis.
- SCHMALZ, CHARLES H.....Mgr. Billings Foundry & Mfg. Co., Billings, Mont.
- SCHUMACHER, WILHELM, Care Allgemeine Brikettierungs Gesellschaft,
Unter den Linden 8, Berlin, W., Germany.
- SEARING, LEWIS, Denver Engineering Works, 30th and Blake Streets, Denver, Colo.
- SUBBALD, ALEXANDER.....Care General Delivery, Joplin, Mo.
- SIELAFF, G. J.....317 Maple St., Reno, Nev.
- SIMONDS, ERNEST H.....805 Crocker Bldg., San Francisco, Cal.
- SIMPSON, J. C., Care E. H. Hunter & Co., 9, Billiter Square, London, E. C., England.
- SMALL, HORATIO L.....452 Newton Ave., Oakland, Cal.
- SMITH, LYON.....534 Humboldt St., Reno, Nev.
- SNEDAKER, EUGENE G.....P. O. Box 604, Goldfield, Nev.
- SNYDER, FREDERICK T.....53 W. Jackson Blvd., Chicago, Ill.
- SPICER, H. N.....Room 2843, Whitehall Bldg., New York, N. Y.
- STAVER, W. H.....684 Fourth Ave., Durango, Colo.
- STEBBINS, H. S.....1300 Leader-News Bldg., Cleveland, Ohio.
- STIMPSON, CHARLES W.....P. O. Box 1014, Salt Lake City, Utah.
- SWANQUIST, G. A.....Divisadero, Salvador.
- TAFF, JOSEPH A.....781 Flood Bldg., San Francisco, Cal.
- TAKEDA, KYOSAKU.....31 Obancho, Yotsuyaku, Toyko, Japan.
- THOMSON, EDWARD.....Hotel Cosmopolita, Guadalajara, Jal., Mexico.
- TRAUTERMAN, CARL J.....832 Colorado Street, Butte, Mont.
- VAN ARSDALE, W. H.....Glenwood Inn, Riverside, Cal.
- VAN LIEW, W. R.....Care F. A. Mattievich & Co., Batoum, Caucasus, Russia.
- WAGNER, WILLIAM HUFF.....3400 Iowa St., Pittsburgh, Pa.
- WEAVER, A. H.....Care Spanish-American Iron Co., Felton, Oriente, Cuba.
- WEINBERG, E. A., Care Charles W. Moore, 5 London Wall Bldgs., Finsbury Circus,
London, E. C., England.
- WESTLAKE, EMORY H.....A. Lewisohn & Sons, 61 Broadway, New York, N. Y.
- WHITTIER, C. C.....2200 Insurance Exchange Bldg., Chicago, Ill.
- WICKS, FRANK R., Butte & Superior Copper Co., P. O. Box 1708, Butte, Mont.
- WILLIS, FRANK G.....415 McPhee Bldg., Denver, Colo.
- WILMOT, H. C.....114 N. 5th St., San Jose, Cal.
- WILSON, E. J.....Commercial Hotel, Thetford Mines, P. Q., Canada.
- WINSOR, F. H., JR.....2940 Broadway, New York, N. Y.
- WRIGHT, PERCY E.....Wilkeson, Wash.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
BLOW, JOHN J.	173 Rodney St., Brooklyn, N. Y.
BOYS, H. R.	Aurora Cons. Min. Co., Aurora, Nev.
CLAGHORN, JAMES L.	P. O. Box 1725, Bisbee, Ariz.
CRARY, CHARLES N.	Kimberly, Nev.
FINLEY, WALTER H.	Logmont, Ky.
GORDON-FIREBRACE, W. E.	812 Salisbury House, London, E. C., England.
GREY, GEORGE R.	54 Ameshoff St., Johannesburg, Transvaal, S. Africa.
HORCASITAS, A. S.	Care Tanawah Gold Min. Co., Sweet Water, Nev.
HYDE, JAMES M.	1041 Shattuck Ave., Berkeley, Cal.
KENNEDY, GEORGE A.	2741 Boulevard F, Denver, Colo.
LANGLEY, SETH S.	601 W. 190th St., New York, N. Y.
MCKIM, JOHN W.	532 Dooly Block, Salt Lake City, Utah.
MAINWARING, H. M. C.	Chillagoe, Ltd., Chillagoe, Queensland, Australia.
MOCINE, JOHN	National Copper Min. Co., Mullan, Idaho.
NOBS, FREDERICK W.	La Leonesa, Matagalpa, Nicaragua.
PORTER, ROBERT S.	Care Fortifications, Culebra, Canal Zone.
RALPH, E. W.	Care Boston Ely Min. Co., Kimberly, Nev.
REVELL, GEORGE E.	P. O. Box 132, Nelson, B. C., Canada.
SALES, A. J.	Giroux Cons. Mines Co., Kimberly, Nev.
SNYDER, FRANK T.	Monadnock Bldg., Chicago, Ill.
STEEL, DONALD	202 Emerson St., Palo Alto, Cal.
SUNDERHAUF, A. L.	Paramaribo, Dutch Guiana.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Mar. 10 to Apr. 10, 1915.

Date of Election.	Name	Date of Decease
1887	*CLERC, FRANK L.	March 12, 1915.
1894	*EVANS, HERBERT A.	April —, 1914.
1891	*OXNAM, T. H.	February 16, 1915.
1901	*REED, DAVID CARLYLE.	January 16, 1915.

*Member.

BIOGRAPHICAL NOTICES

August Christian, a life member of the Institute since 1899, died at Butte on Nov. 19, 1914. He was born at Altstadt, province of Wiesbaden, Germany, on July 10, 1853, and studied engineering at Freiberg. After serving in the employ of the German government on railroad surveys for six years he came to the United States in 1883. After a few months in Cincinnati he was engaged in railroad construction work on the Oregon Short Line in Idaho and Montana. Returning to Cincinnati, he married and secured employment in the city engineering department, where he remained until 1887. He was then again connected with the engineering staff of the Union Pacific for two years in Kansas, Colorado, and Indian Territory. In 1889 he moved from Colway, Kan., to Anaconda, Mont., as engineer for the Montana Union Railway, building lines from Butte to Garrison and from Stewart to Anaconda. In 1891 he moved to Butte as engineer in charge of the mining operations of the late Marcus Daly, and retained his connection with the mines of Butte, as chief engineer for the Anaconda Copper Mining Co., until his death.

Of sturdy physique, cheerful demeanor, industrious habits, and unobtrusive, even modest, bearing, August Christian was one of the few men

who enjoyed the implicit confidence of Marcus Daly during the trying days of the development of the great copper mines at Butte. Thoroughly competent and reliable under all circumstances, always ready to extend a helping hand to his friends, and asking nothing in return but their continued friendship, he displayed qualities of mind and heart which won for him success in his labors and acknowledged leadership in his profession. More than this, his example has been a stimulus to all those who served under him and were associated with him. He was, although not widely known, an honored member of the Institute and of the engineering brotherhood.—H. V. W.

Samuel Gibson McNulty was born Dec. 24, 1862, near Monterey, Highland County, Va. After preparatory studies at Virginia schools, he entered the Virginia Agricultural and Mechanical College at Blacksburg, from which he was graduated in 1888 as Bachelor of Science in Engineering, and in 1889, after a post-graduate course, as Civil Engineer. After graduation, he entered the service of the Norfolk & Western Railway Co., first as chainman and rodman in the construction of the Clinch Valley division, afterward in the coal department on the newly opened Ohio extension, and finally as Superintendent on the Kenova division. This occupied him until 1896, when he became Chief Engineer of the Playa de Oro Gold Mining Co. in Ecuador, South America, where he spent four years in constructing canals, tunnels, flumes and dams for the hydraulic mining operations of that company. Returning from this work, he practiced in West Virginia for a time as a civil and mining engineer, after which he became resident engineer in the construction of the Big Sandy Extension, from Naugatuck to Kenova, of the Norfolk & Western Railway. On the completion of this work, he undertook for himself and his associates the opening of the mines and installation of the plant at Rose Siding, W. Va., of the Thacker Coal Mining Co., of which he was until his death Vice-President and General Manager. Mr. McNulty died Mar. 13, 1915, at Hot Springs, Ark., where he had been a patient for two months, vainly seeking recovery from Bright's disease. His funeral, at Staunton, Va., was attended by many persons from the coal field in which his latest work had been done, as well as by numerous friends to whom his upright and amiable character and active benevolence had endeared him, while his executive ability had won their admiration and esteem. He became a member of this Institute in 1902.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Application of the Apex Law at Wardner, Idaho

BY FRED T. GREENE, BUTTE, MONT.

(San Francisco Meeting, September, 1915)

Most of the recent discussion of the mineral land law published in the *Transactions* is in the abstract—an exception being Mr. Goodale's paper, *The Apex Law in the Drumlummon Controversy*,¹ which illustrates in concrete instances many of the practical disadvantages of what Dr. Raymond has called "The Law of the Apex." The present paper, it is hoped, may serve a similar purpose, emphasizing the necessity for a change in our mining laws, and bringing home to mining engineers some of the defects and dangers of our present code.

There are many storm centers of "apex litigation" in the Rocky Mountains; and not the least of them is the Wardner camp, where such litigation has been almost continuous since 1892, and the end is not yet.

In their decisions, the courts have accepted the claims of the litigants that the "Wardner vein" is a broad zone varying from 250 to 500 ft. in width at the surface. This is the simplest theory which could have been advanced, and probably decreased, rather than augmented, the number of contests; in fact, every one of the claim owners seemed to feel that the greatest advantage would accrue to him if this theory were adhered to. The dominant structural feature of the Wardner vein is a persistent fault, striking N. 40° W., dipping about 40° to the southwest and called the "Wardner foot wall." Fig. 1 is a composite of two adjacent cross-sections of the vein and is a fair representation of the relative size and shape of the orebodies and their relation to the "Wardner foot wall." The country rock is a comparatively thin-bedded sericitic quartzite which has an average strike of N. 80° W. and dips at high angles to the north and south.

The movement on the Wardner foot wall is apparently normal, and there has been more or less brecciation on either side of it; but as the moving mass was that above the foot wall, the brecciation has been more complete on that side, and has extended farther from the plane of movement. Many minor faults, generally of insignificant extent, resulted from the movement along the Wardner foot wall. Most of these were

¹ *Trans.*, xlviii, 328 to 341 (1914).

pre-mineral, although some post-mineral dislocations can be observed. The period of mineralization must have been very long, and these minor faults and the bedding of the quartzite largely controlled the shape and position of the orebodies. The average strike of these minor faults is N. 14° W. with a predominating dip of about 60° to the west. The ore seams have an average strike of approximately N. 50° W., which is practically a mean between the bedding planes and the minor faults.

The orebodies rarely rest immediately upon the Wardner foot wall,

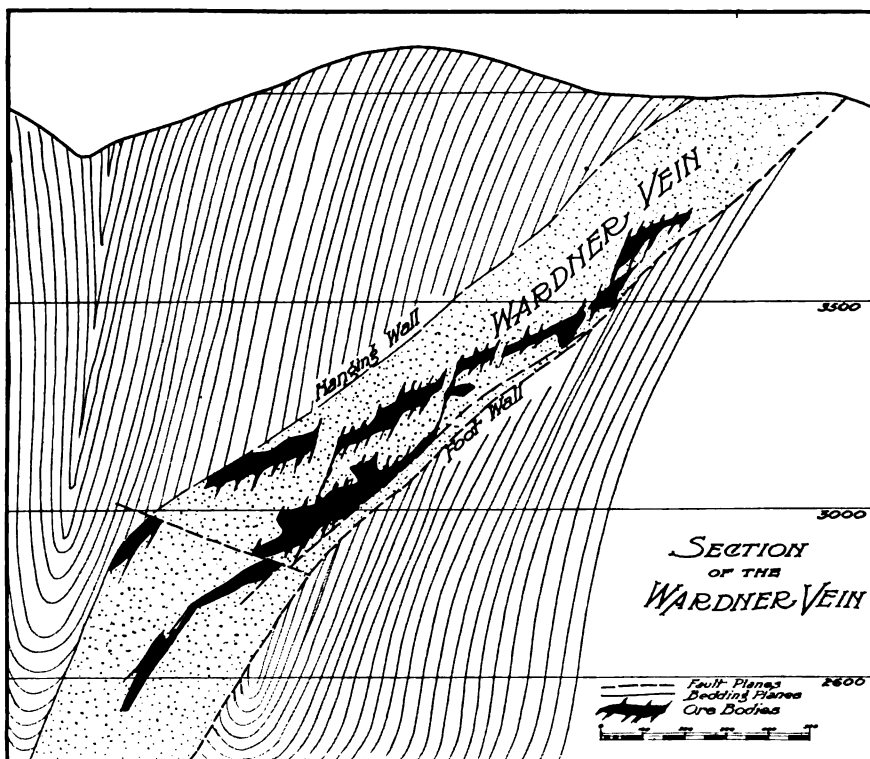


FIG. 1.—SECTION OF THE WARDNER VEIN.

but depart from it at sharply acute angles both along the strike and dip. By reason of extensive alteration and oxidation along bedding planes, this departure of the orebodies from the Wardner foot wall was even more pronounced at the surface; so that it is not at all surprising that the discoverers, in the necessary hurry of marking their boundaries, placed their claims more nearly parallel to the bedding than to the foot wall. It is to this error that most of the litigation at Wardner has been due.

There are four fundamental conclusions of law regarding the extra-lateral right of the discovery vein.

1. When the vein passes through both end lines. This condition is, of course, covered by the text of the law.

2. When the apex of the vein crosses one end line and stops before reaching the other. Here again the end lines define the right.²

3. When the apex enters the claim through one end line and departs across either side line. In this case the extra-lateral right is defined by two planes, one through that end line, and the other through the point of departure across the side line, and parallel to the end lines.³

4. When the apex of the vein crosses both side lines. Here, providing the side lines are parallel or converge in the direction of the dip, the side lines become end lines and the extra-lateral right is contained between vertical planes through these side lines extended in their own direction.⁴

This feature of the apex law has been analyzed by Dr. Raymond,⁵ who introduces his discussion of it by remarking:

"There is apparently no end to the doubts, inconsistencies and absurdities in which the courts of our mining States and Territories are involved in their attempts to apply to conditions of ever-increasing complexity the provisions of our anomalous mining law."

This language was prophetic and would be none the less so were it used to-day.

In no mining camp has this question of side-line crossings been more frequently raised than at Wardner. Had individual claims been maintained, the accompanying sketch, Fig. 2, based on the conclusions of one of the ablest mining attorneys in the Rocky Mountain States, would define the ownerships in that camp. Had not the controversy between the two companies operating along this vein been settled out of court, it is easy to conceive that the litigation over extra-lateral rights would have been interminable.

Twenty years of litigation established the rights of the following claims as shown in Fig. 2: Lackawana, Sullivan, Bunker Hill, Stemwinder, Emma, Last Chance, Tyler, King Fraction, Viola, and San Carlos; but the rights of the Phil Sheridan, Republican Fraction, Cheyenne, Lincoln, Sims Fraction, and Overlap remained to be determined.

The Phil Sheridan right would probably be settled more or less automatically; but, because of the immense orebodies between the Phil Sheridan and Tyler extra-lateral right, suits based on the Republican Fraction, Sims Fraction, and Overlap would have been bitterly contested.

An early effort to acquire the Republican Fraction right was fruitless; but the "doubts, inconsistencies and absurdities" of the apex law are so

² Carson Co. vs. North Star Co., 73 Fed., p. 597.

³ Fitzgerald Co. vs. Last Chance, 171 U. S., p. 55.

⁴ Flagstaff Silver Mining Co. vs. Tarbel, 98 U. S., p. 463; Argentine Co. vs. Terrible Co., 122 U. S., p. 478; Last Chance Co. vs. Tyler Co., 157 U. S., p. 683.

⁵ End-Lines and Side-Lines in the U. S. Mining Law, *Trans.*, xvii, 787 (1888-89).

manifold that it is not only possible but even probable that a second attempt might have succeeded. However, if such second attempt did fail there would be the rights of the Sims Fraction and Overlap to be taken into consideration. To the layman it might seem absurd to assume that either of these claims could possess rights to an orebody more than a half mile away and 1,700 ft. below the surface; but it is owing to a decision of the Department of the Interior that such claims were located.

The papers in this case are not accessible to me, but the history of it is this: Immediately west of the Viola and San Carlos was the Oakland, which overlapped both the former claims. The Viola was patented as shown; but when the San Carlos was patented the conflicting area of the Oakland was excluded. Later, when patent to the Oakland was applied for, the Department decided that the Oakland could not claim more than the total length of "free apex," and the east end line of this claim had to be drawn in, with the result that two fractions were left, which were located and patented as the Overlap and the Sims Fraction.

The Cheyenne would have given rise to a case very similar to the Drumlummon.

The remaining claim, the Lincoln, was located to acquire an open right immediately west of the King Fraction. This location, like the Copper Trust of Butte, was made on some widely separated fractions. But, while the claims of the Copper Trust were denied by the Supreme Court of Montana, that decision in no wise settled the rights claimed by the Lincoln, and the following opinion written by two eminent Colorado lawyers seems pertinent. They say:

"It has been said that hard cases make bad law. This seems to be particularly applicable to the Copper Trust case. The only surface open to location, and therefore appropriated by the Copper Trust location, was two exceedingly small triangles at the extreme easterly end of the location, only one of which, and that only for a slight distance, contained any part of the apex of the vein. The orebodies claimed by the Copper Trust under its extra-lateral right lay to the southwest of the Smokestack claim, and therefore within the extra-lateral right claimed by reference to the apex of the vein lying in the extreme westerly portion of the Copper Trust location. The

Dates of Location of Mining Rights on the Wardner Vein (see Fig. 2).

Bunker Hill.....	Sept. 10, 1885.	Republican Fraction..	Nov. 1, 1885.
Phil Sheridan.....	Sept. 16, 1885.	Amended.....	May 11, 1886.
Last Chance.....	Sept. 17, 1885.	Viola.....	Feb. 20, 1886.
Stemwinder.....	Sept. 17, 1885.	San Carlos.....	Apr. 23, 1886.
Amended.....	May 25, 1887.	Amended.....	May 11, 1886.
Emma.....	Sept. 17, 1885.	Cheyenne.....	Oct. 25, 1891.
Lackawana.....	Sept. 18, 1885.	King Fraction.....	June 22, 1898.
Tyler.....	Sept. 20, 1885.	Sims.....	Jan. 8, 1901.
Sullivan.....	Oct. 2, 1885.	Overlap.....	Apr. 9, 1901.
Lincoln.....	Sept. 8, 1905.		

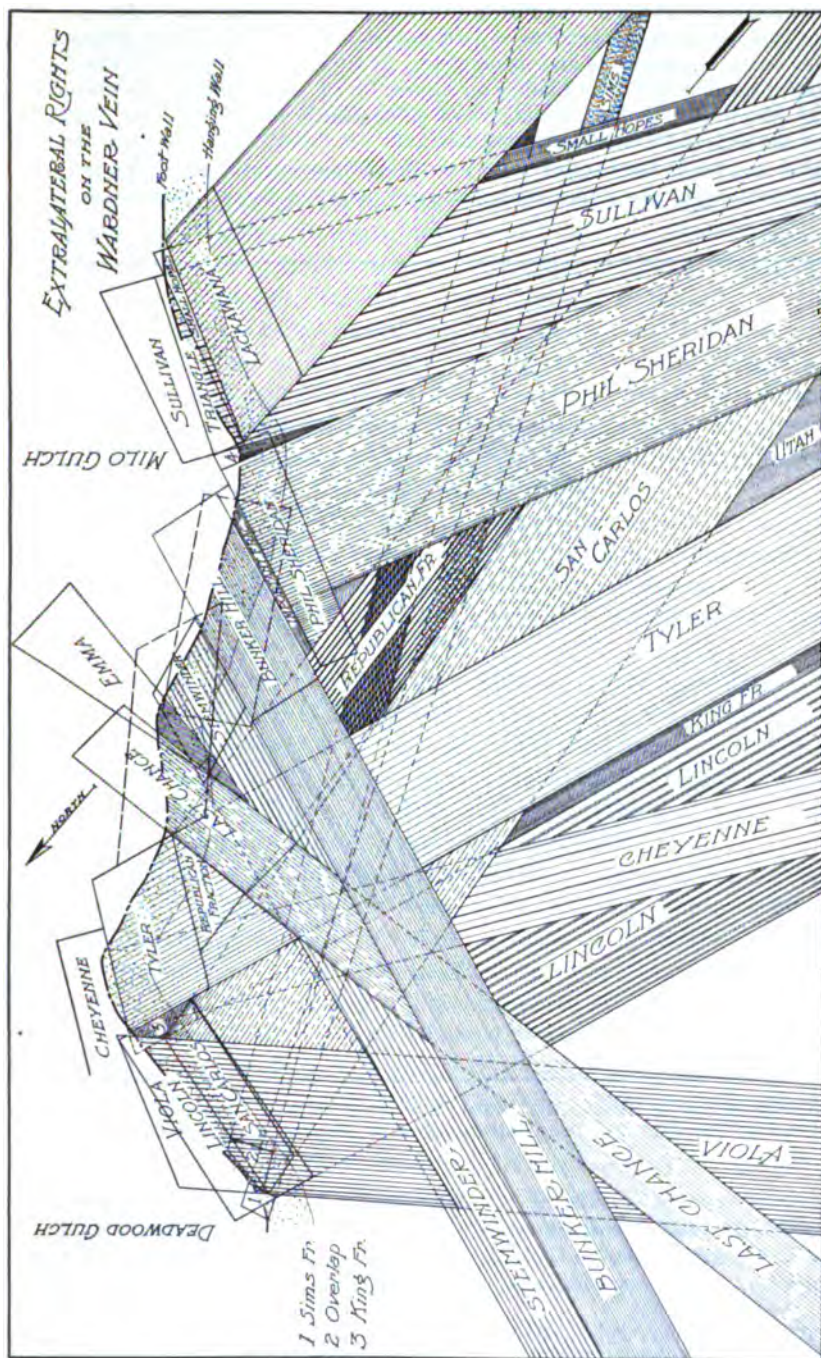


FIG. 2.—EXTRA-LATERAL RIGHTS ON THE WARDNER VEIN. SCALE, 1 IN. = 1,333 FT.
(See tabulation of dates of location on opposite page.)

latter complained of was an unusually broad order for inspection and survey. The logic of the opinion is not persuasive. Its analysis of the Del Monte decision, as it seems to us, is inaccurate and incomplete. It cites as an authority (65 Pac., 1025) sec. 780 of the first edition of Lindley on Mines, a book issued prior to the decision in the Del Monte case and expressing views subsequently modified by the author in view of the Supreme Court's later decision.

"It is asserted in the Copper Trust case that in any event the Del Monte case is not applicable when the senior claim has gone to patent before the location of the

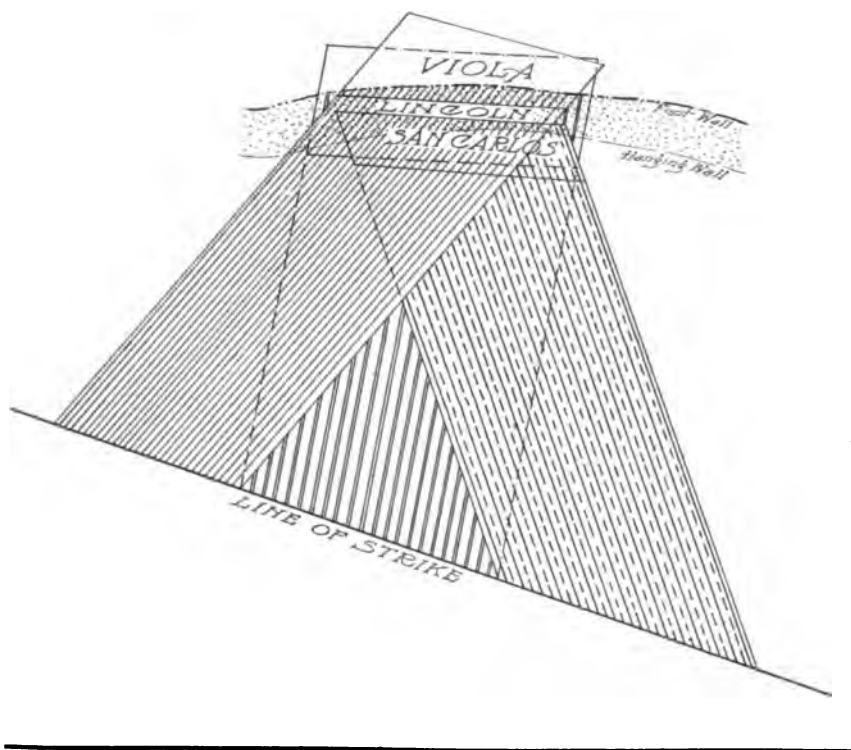


FIG. 3.—CONDITIONS RESULTING FROM A HYPOTHETICAL SEPARATION OF THE SAN CARLOS AND VIOLA MINING RIGHTS. SCALE, 1 IN. = 1,000 FT.

junior claim, but this is certainly in conflict with the authorities and erroneous on principle. The land department takes the opposite view (*The Hidee Gold Mining Co.*, 30 L. D., 420). The contrary view is certainly expressed by the Circuit Court of Appeals in the *Stemwinder* cases, and Judge Lindley says: (*I Lindley on Mines*, 1 ed., Sec. 363a, p. 657.)

"There is no reason which could be urged in support of permitting junior locators to rely on the lines of their claims across prior unpatented claims which could not be invoked in behalf of a similar doctrine as applied to patented claims of all classes."

Whether this opinion be correct or not, the fact that it is the expression of reputable attorneys is a guarantee that it would be entertained by a court of law; and a long and expensive law suit would result.

It is easily conceivable that the San Carlos and Viola might have been separated by ten or more feet, and the condition shown in Fig. 3 would have existed. In this event there is little doubt but that the Lincoln would have had extra-lateral rights as shown and thus a claim of 1,500 ft. along the apex of the Wardner lode would have controlled rights to 4,100 ft. along the strike, at a depth of about 1,500 ft. (One of the "absurdities," no doubt, which Dr. Raymond had in mind.)

In the earliest of the many valuable papers contributed by Dr. Raymond to the subject of American mining law,⁶ is a map (p. 436) which bears a remarkable resemblance to Fig. 2 herewith given; and this coincidence is the more remarkable, since Dr. Raymond's paper was presented at the Troy, N. Y., meeting in October, 1883, more than two years before the first discovery was made in the Wardner district.

The litigation at Wardner affords not only a convincing example of the inadequacy of "the law of the apex," but clearly demonstrates that this method of acquiring the right to a mining location, based on a legitimate discovery, frequently fails to secure to the discoverer the vein he sought to locate.

The operators who have had any experience with suits arising from the law of extra-lateral right agree that the cost of such litigation is an irksome burden, and often, rather than incur the costs of such a suit, or venture into a court of law which is at variance with itself, they compromise with their neighbors, even though convinced that they are not getting all to which they believe themselves entitled.

On the other hand the lawyer has to rely upon the geologist for his facts as to structure, etc. It is conceivable that a mining geologist may become obsessed with a theory which he will apply to certain conditions, entirely neglecting some important fact which might destroy the logic and admissibility of his theory. The idealist will claim that any mistake of this kind would be eliminated before a case depending upon such a theory could come to trial; but even if this were so, there has been a great injustice done when it is possible to bring suit upon such evidence, and enjoin the defendant company from working parts of its mine, if it develops later, when the suit is dismissed, that the defendant company can get no commensurable recompense for the damage done it.

Naturally, the law is not exclusively responsible for these opportunities of the unscrupulous to use it for irreparable injury to others; but the opportunities will always exist, and the mischievous and preposterous predicament will be possible, so long as we are bound by the law of extra-lateral right. Dr. Raymond sums up the question very pertinently:

⁶ The Law of the Apex, *Trans.*, xii, 367 (1883-84).

"If all mining properties presented this beautiful simplicity of structure (ideal locations on ideal veins), and all mining locators exhibited a corresponding simplicity of purpose, the application of the law would be easy, but the naïveté of the statute fares badly between the freaks of nature and the tricks of man."

With the development of the science of ore deposits, our belief in the "freaks of nature" is rapidly fading, and many of the phenomena styled "freaks" are explainable. However, the other horn of the dilemma, while as easily explained as these alleged freaks, is just as hard to contend with as formerly, and under the existing mining laws there is small chance of circumventing these "tricks of man."

The law works badly in another way. Its language is decidedly untechnical; and ever since the first apex suit the courts and witnesses have tried to provide technical definitions for many of its terms and have succeeded only in clouding the meaning intended by the framers of the law. Were Milton a member of the Committee on Mining Law he would probably apply his own language to the law of the apex:

"Chaos umpire sits,
And by decision more embroils the fray
By which he reigns; next him high arbiter,
Chance governs all."

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The Mellen Rod-Casting Machine

BY R. C. PATTERSON, JR., NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

In view of the circumstance that very few important changes have been made within the last 15 or 20 years in the equipment of rod and wire mills, the description of a new process introduced by Grenville Mellen, of Llewellyn Park, N. J., to take the place of the present laborious system of producing rods of lead, zinc, brass, copper, aluminum, etc., may be interesting to members of the Institute. This new process consists in the production of cast rods at one operation in a small continuous casting machine.

The hot liquid metal is transferred from the melting crucibles directly into an endless chain of mold blocks in the machine, where solidification takes place, and the rod comes out continuously in a solid form at one end as long as the molten metal is supplied. The operation of these mold blocks so as to produce a solid rod of uniform structure constitutes an important part of Mr. Mellen's invention.

The machine (Figs. 1 and 2), 12 ft. in height over all, 2 by 3 ft. in horizontal cross-sectional area, and 6,000 lb. in weight, has a framework of cast iron, holding in position two endless chains of mold blocks, which are made in sections and join in center alignment. The mold orifice is made up of these mold sections, into which the liquid metal flows.

The molding chains are composed of steel blocks, grooved on one face to form the molding cavity, and linked together with flexible joints. Each block is carried on four rollers which guide the chain around the end sprockets and carry it in its course through the machine.

The accuracy of alignment requisite to the production of a perfect rod in the mold groove formed by these sections is secured by careful machine work, and by building the four ways, down which the alignment takes place, so that two of the sides are fixed permanently, while the opposite sides are held to their position by spring pressure. The guides carrying the molds, while in casting position, are water-cooled square tubes.

The length of this machine is somewhat indeterminate. A certain amount of both time and cooling surface is required to solidify the metal

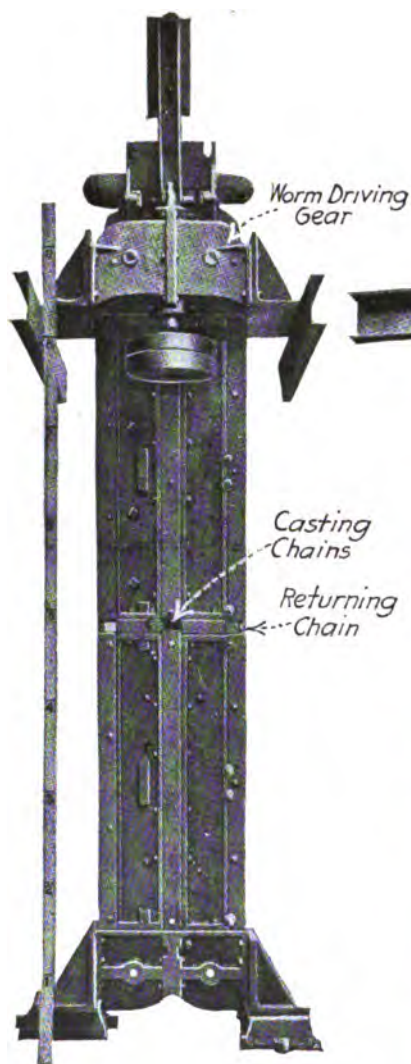


FIG. 1.

FIG. 1.—FRONT VIEW OF VERTICAL TYPE OF MELLEN ROD MACHINE. THIS MACHINE CASTS BRASS, COPPER, ZINC, ETC.

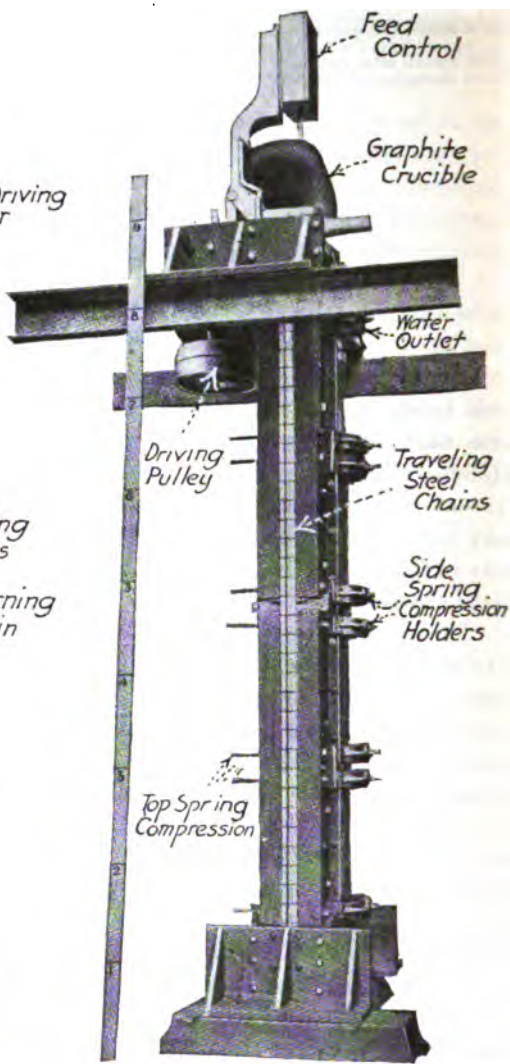


FIG. 2.

FIG. 2.—SIDE VIEW OF VERTICAL TYPE, SHOWING CASTING CHAIN WHICH TRAVELS UPWARD TO MEET IN A COMMON CENTER THE OPPOSITE CHAIN, THESE TWO UNITING TO FORM A SOLID STEEL MOLD INTO WHICH THE MOLTEN METAL IS CAST THROUGH A SILICA TUBE EMERGING FROM A GRAPHITE CRUCIBLE.

in the mold; and one may either use a long molding chain and run it fast, or a shorter chain and run it correspondingly slower. We have found, however, that with 8 ft. length between centers for the chains a casting speed of 25 ft. per minute for $\frac{7}{8}$ -in. brass rod will not cause the molding blocks to rise in temperature above 450° F.

Experiments have shown that in the casting of bronze mixtures, the pair of molding blocks should have about six times as much sectional area as the cast rod, to allow a reasonable temperature difference for the rapid removal of the heat.

The first machines designed for this process of continuous casting were vertical; but the desire of practical rod-mill men to operate with the molding groove horizontal led Mr. Mellen to spend considerable time in experimental improvements of the mechanical details, in order to secure a maximum efficiency for the horizontal machine. In this attempt, numerous troubles and intricate problems were encountered. The machine would always cast rods; but its successful operation in a horizontal position required more skill than that of an ordinary laborer. After much experiment the axis was first made slightly inclined; but finally the vertical position was readopted; and in this form one laborer, with a little preliminary instruction, can manage the machine without difficulty.

The chief hindrances to horizontal operation were: (1) the difficulty of completely filling the molds, and (2) the circumstance that, in working horizontally, the lower chain did the greater part of the work and became after a time excessively heated, while the upper chain remained relatively cool. This caused an irregular structure both in the cast rod and in the drawn product resulting therefrom; and as a result the required strength tests were not satisfactory.

In the vertical arrangement these difficulties automatically cured themselves since the metal completely filled the mold, with uniform contact all round, thus causing each portion of the chain to do its work, and giving a symmetrical structure to both rod and resulting wire.

The importance of making all of the chain work all the time is shown by the fact that in the vertical type of machine the chain can be considerably shortened, and yet the machine is capable of running at higher speed than when arranged horizontally. This vertical arrangement, however, necessitates operation on two levels.

The flow of metal into the machine is controlled by an electrically operated automatic device, which adjusts the feed to the speed at which the machine is operated. If for any reason the machine should stop the flow would be automatically shut off.

A safety device is incorporated into the drive, so that if any foreign material clogs the chain a safety pin is sheared, thus protecting the mechanism from injury.

The rods are delivered from the machine immediately to the die of the bull block, where they are drawn down to suit particular orders.

This new method of manufacturing rods does away with the following steps of the old system:

1. Casting the wire bar.
2. Handling the wire bar from the molds.

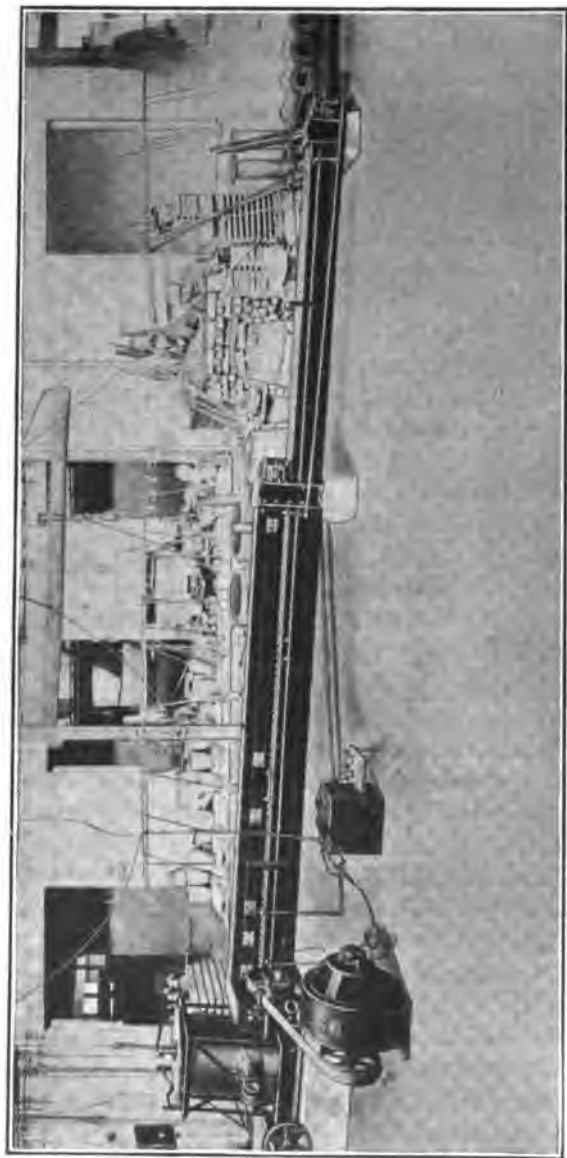


FIG. 3.—THE HORIZONTAL TYPE OF MELLEN ROD MACHINE FOR LEAD AND ALUMINUM.

3. Rehandling the wire bar to and from the reheating furnace.
4. Reheating the wire bar.
5. Rolling the wire bar.

It also eliminates the loss from oxide scaling during heating and rolling. These economies involve, of course, the saving of much capital, now invested in heating furnaces, rolling mills, power plants, land, and buildings. Since the power required to run this machine is only about 5 hp., the cost of large engines, boilers, coal bins, etc., is practically done away with.

Another advantage is the elimination of danger to human life. The apparatus is completely inclosed, so that the operator is protected from injury. Finally, the labor cost of this process is but 5 per cent. of that of the old method, the caster being the only workman required.

A later development of this continuous casting machine is the machine (Fig. 3) casting rods of lead and soft-metal alloys, $\frac{1}{4}$ in. and upward in diameter. This is the first step in the manufacture of shrapnel bullets; and the machine will cast per hour enough lead rod for more than 200,000 bullets. The only motive power required is that which is necessary for overcoming the friction resisting the motion of the traveling mold chains, which is in the neighborhood of from 2 to 3 hp.

This lead machine is practically the same as that used for brass with the exception that it is operated in a horizontal plane. Also, the horsepower is slightly decreased, and the machine is a few feet longer.

In two drawings, the area of brass rod is reduced about 27 per cent. without annealing; but it must be realized that the area reduction and pull are variables depending upon the mixture cast. An 80-hp. motor is used in handling the $1\frac{1}{4}$ -in. rod.

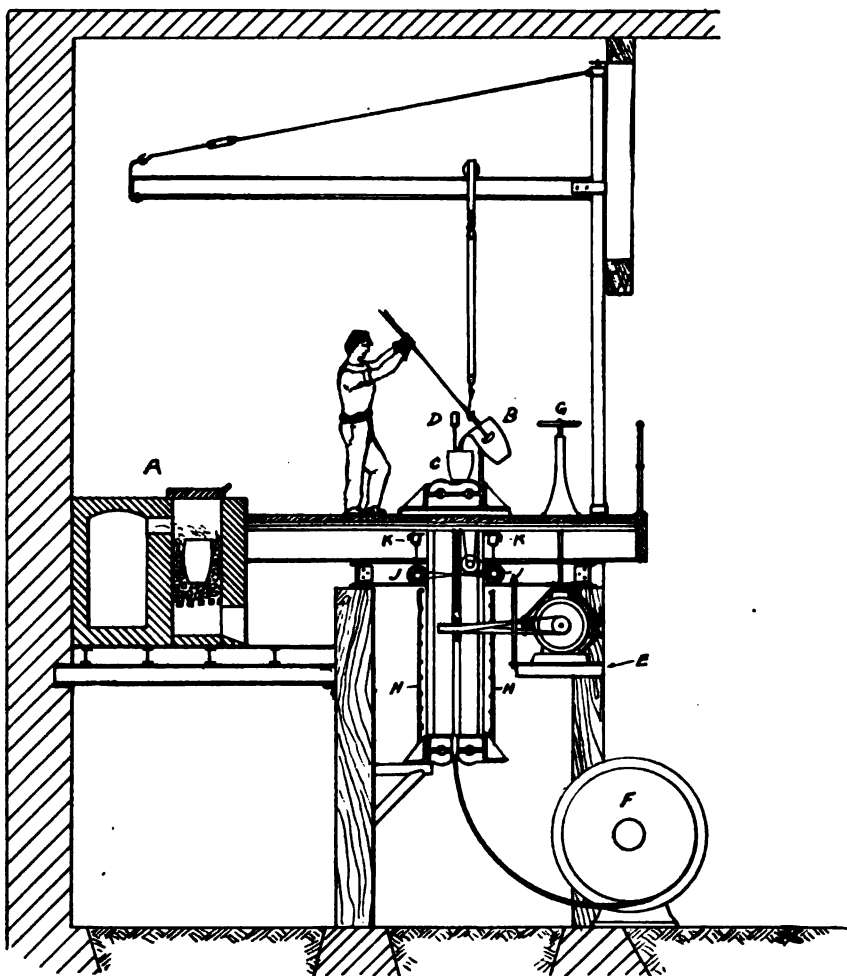
The continuous casting machine is also applicable to the production of aluminum and zinc rod. Aluminum rods are produced with diameters ranging from $\frac{3}{4}$ in. upward.

The casting-shop arrangement is shown by Fig. 4. This may be considered a cross-section of the shop with a long line of melting furnaces in parallel to the one shown, all of which supply metal to one casting machine. The melting furnace *A* is one of 40, the crucibles of which have a capacity of 220 lb. each with six heats per 10 hr. All of these are considered just sufficient, when operating at full capacity, to satisfy one casting machine in 10 hr. The feed basin *C* consists of a graphite crucible and has a capacity of 194 lb. The automatic feed-control device *D* consists of a solenoid magnet operating in a battery circuit, one terminal of which is grounded on the machine and the other terminal attached to a graphite point projecting into the mold orifice. When the metal in the mold rises to this graphite terminal it closes this circuit and the magnet is energized. This magnet operates upon a beam to partly close a carbon valve controlling the inlet of metal into the mold bore. When the rod that is being cast into the molds is carried below this graphite tip, the circuit is broken and the valve lifts, allowing more metal to enter the machine.

The variable-speed motor *E* is used to drive the machine. The varia-

tion in speed is necessary to enable the machine to be operated on different brass mixtures, some of which flow more freely than others. This speed is suitably adjusted by the operator at the speed-control wheel *G*.

The reeling device *F* takes the rod delivered from the machine and rolls it into bundles to facilitate its further handling. The cooling water



- A. Melting Furnace
- B. Melting Crucible
- C. Feed Basin
- D. Automatic Feed Control
- E. Variable-Speed Motor

- F. Reeling Device
- G. Speed-Control Wheel
- H. Cooling Water Jets
- J. Mold-Cleaning Brushes
- K. Air-Brush Mold Facer

FIG. 4.—ARRANGEMENT OF CASTING SHOP.

jets *H* are a series of pipes arranged opposite the faces of the returning molds through which a series of streams of water are played directly upon the mold faces for cooling purposes. These streams are not turned on

until the molds have reached a temperature sufficient to vaporize this water. In practice the molds appear perfectly dry a few inches above the uppermost jet. Above these water jets are noted the mold-cleaning brushes *J*. These, which are made of steel, revolve by a chain mechanism and serve to clean any deposit from the surface of the mold and prepare it to receive the necessary facing, which is applied in liquid form by means of air brushes *K*.

The machine's operating cost is less than 30c. per ton of molten metal into the form of rods.

U. S. and foreign patents have been secured for this invention.

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The Electric Furnace in the Foundry

BY WILLIAM G. KRANZ,* SHARON, PA.

(San Francisco Meeting, September, 1915)

THE increased service demands on some of the products of the National Malleable Castings Co. prompted it about eight years ago to investigate the electric furnace, both in America and in Europe. The process had already been sufficiently developed in Europe to lead us to believe that the electric furnace would most nearly meet our requirements.

After further investigation and consultation with the highest authorities on the subject, we decided upon the Heroult type as the simplest and most practicable.

In 1910, we built a furnace of this type, of 300 lb. capacity, for experimental purposes, and although it was crudely constructed the quality of the material we were able to make justified the installation of a 6-ton Heroult furnace in 1912.

This furnace has three electrodes, 17 in. in diameter, uses three-phase, 60-cycle current, and is equipped with Thury regulators, which give us good regulation and a uniform load on the line even when cold melting. The power factor of the furnace is 94 per cent.

Since July, 1912, the furnace has been in continuous operation, producing up to the present time over 20,000 tons of both carbon and alloy steels of varying analyses and of exceptional quality. Both cold and hot-metal charges have been used, with a power consumption of about 150 kw-hr. per ton in the case of hot-metal charges and from 500 to 600 kw-hr. per ton in the case of cold charges.

The electric energy consumed varies according to the final analysis of the steel and the amount of refining required. Too much cannot be said about the refining possibilities of the electric furnace, for no other method of steel manufacture can compete with the electric in this respect.

In proper operation lies the whole secret of success. That steel has been made electrically means nothing, for when the furnace is operated under oxidizing conditions the quality is no better than that of open-

* Non-member. The National Malleable Castings Co., Sharon, Pa.

hearth material; but when properly made, electric steel is as good as that made in the crucible, and very much less expensive. The operation of the furnace is simple and the resultant composition is scientifically accurate. I might cite an example in this connection:

On a heat of steel treated under a slag of high silica content some metallic aluminum was added in order to reduce the silicon from the slag in accordance with the following chemical equation: $3\text{SiO}_2 + 4\text{Al} = 3\text{Si} + 2\text{Al}_2\text{O}_3$. Upon final analysis it was found that the amount reduced was theoretically correct in accordance with the above equation.

The most important of the many advantages of electric steel castings over those made by the ordinary processes are briefly summarized below, and it should be kept in mind that these apply not only to carbon steels but to the alloy steels as well.

1. Absence of segregation, elimination of oxides, and absolute uniformity of composition regardless of atmospheric conditions which affect open-hearth furnaces.

2. Almost entire elimination of sulphur is possible (an important consideration in steel castings) and complete control of the other elements.

3. Great tenacity, giving ability to withstand much more abuse and fatigue without rupture.

4. High ratio of elastic limit to ultimate strength.

5. A more ready response to heat treatment and with much more uniform results.

6. Perfect control of pouring temperature, combined with ability to obtain very hot metal, so that light and intricate shapes are readily cast.

Let us take up the advantages of the more important of these qualities of electric steel, and investigate them more fully.

First.—Absence of segregation and oxides has been firmly established by a great many investigations in our chemical and microscopical laboratories. As a more practical proof of the elimination of oxides, we know that additions of any of the ferro-alloys to the bath will be found alloyed with the steel in their theoretical amounts. This is even the case with elements, such as aluminum and titanium, which are so susceptible to oxidation, proving conclusively in our minds that there could be no oxygen present in the steel. Leading authorities have agreed for some time that the electric steel furnace is the one means of preventing segregation.

W. R. Walker, in May, 1912, read a paper on Electric Steel Rails before the American Iron and Steel Institute, in which he stated that "Ingots of even eight tons had been produced electrically which were practically free from segregation."

Second.—We know of no authority who doubts the practical elimination of sulphur in electric-furnace operation. We have repeatedly

reduced the sulphur to a trace in irons containing as high as 0.30 sulphur. The rapidity with which this reduction takes place depends somewhat upon the carbon content of the material, the sulphur reducing very much more rapidly in the higher carbon materials.

Third.—To illustrate the great tenacity of electric steel we submit (Fig. 1) exact reproductions (reduced) from two tests of two similarly treated specimens of steel, which are typical of a large number made on an Upton-Lewis toughness testing machine. One of these is electric and the other open hearth, of almost identical analysis, as shown below. The marked superiority of the electric steel specimen, especially as to its tenacity, is shown in the following tabulations:

Basic open-hearth steel (annealed). 213 cycles.

Electric steel (annealed). 345 cycles.

FIG. 1.—FATIGUE TESTS.

Electric Steel

C, 0.24; Mn, 0.52; Si, 0.25; P, 0.010; S, 0.019.

Elastic limit.....	36,400 lb.
Ultimate strength.....	65,300 lb.
Elongation.....	36 per cent.
Reduction.....	55 per cent.
Toughness test (fatigue) to break.....	345 cycles

Basic Open-Hearth Steel

C, 0.23; Mn, 0.53; Si, 0.24; P, 0.011; S, 0.038.

Elastic limit.....	34,800 lb.
Ultimate strength.....	63,000 lb.
Elongation.....	29½ per cent.
Reduction.....	35 per cent.
Toughness test (fatigue) to break.....	213 cycles

Fifth.—During the past year we have developed an electric steel having remarkable physical qualities after heat treatment. This was accomplished without resorting to any of the high-priced alloys. The steel, when subjected to shock or static pull, will stand from four to five times as much stress without distortion as the ordinary open-hearth product. This assertion is not based on a few tests, but on over 3,000 made up to the present time.

We have made a great many similar tests of heat-treated open-hearth material, and have invariably found that in certain specimens

the physical properties are impaired by the treatment rather than benefited. This is not, however, the case with the electric-furnace product. The old saying that "In order to make good bread you must start with good dough," we believe applies to the steel industry as well.

CONCLUSIONS

The greatest advantage of the electric-furnace process over all others is its uniformity of product. The open hearth under certain conditions will, we know, produce steel of very good quality, but there are so many contingencies, such as atmospheric conditions, stack draft, fuel and furnace variations, beyond the control of the operator, that absolutely uniform results are impossible.

Electric-furnace products are looked upon by some with skepticism, due to those who have adopted the process thinking it a panacea for all of their ills, and who have furnished the trade with products not worthy of the name. The lack of knowledge and the inexperience of the operator should in no way condemn the process. The production of perfect castings does not entirely depend upon the quality of the material of which they are made, but upon the foundry practice to a very great extent. Frequently excellent material is ruined by bad foundry practice. Nevertheless we feel that the electric furnace, with its perfect control of composition and temperature, fills a long-felt want in the industry.

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Mill and Cyanide Plant of Chiksan Mines, Korea

BY CHARLES W. DE WITT, SEIKWAN, KOREA

(San Francisco Meeting, September, 1915)

THE ore treated at the reduction plant (called Yangdei) of the Chiksan Mining Co., Korea, is brought from four of the company mines, and from the small tribute mines. The largest shipments come from Sajunkohl mine, situated about a mile and a half from the mill (Fig. 1). The ore



FIG. 1.—SAJUNKOHL MINE OF THE CHIKSAN MINING CO., KOREA.

is quartz of medium hardness. The average assay value in gold is \$6.95 per ton. At the mine the ore is put into bins, and is then trammed over an 8 per cent. grade to the tippie bins above the mill, where the ore is drawn through the mill holes as desired. The capacity of the tippie bins is about 3,500 tons.

Ore from Moonsukohl, Chung Tarrie, Yang Chun, and the tribute mines is brought in by bulls and bull wagons. The bulls carry from 350 to 800 lb.; the average wagon load is about 1,200 lb. (Fig. 2). One shovelful of ore out of every ten is taken as a sample. Considerable ore

is brought in from the outside mines, and this method of sampling is used to avoid mixing the lots. However, a series of bins will be erected in the near future and an automatic sampling device installed so that the ore can be handled to better advantage—and a much safer sample secured, with less room required.

In January, 1907, ground was broken for the first two Nissen stamps, the first ore was treated in February, 1908, and the mill was in continuous operation until August of that year. The cleanup for this run was \$6,024. Owing to shortage of supplies and mill trouble the plant was closed until May, 1909, but since that time has been in continuous operation.

The output from February, 1908, to Oct. 31, 1909, was \$11,113. For the six months ended Oct. 31, 1909, 2,840 tons of ore were milled, in 159



FIG. 2.—SORTING ORE AND LOADING BULLS AT ONE OF THE CHIKSAN MINES.

milling days (this includes 11 days that two extra Nissen stamps were running). The recovery from bullion for this term was \$5,090.

In June, 1909, two more Nissen stamps were added to the mill. The mill then ran until July, 1911, when it was found advisable to increase the crushing capacity, so two batteries of five 1,250-lb. stamps each were added.

In April, 1912, one more battery of five stamps was added, and in June of the same year four batteries of the same type were added. This made a total of 39 stamps, but owing to the wear and tear on the Nissen stamps it was found necessary to replace them. In January, 1915, the Nissen stamps were taken out and are being replaced by two 5-stamp batteries. Since 1907 our plant has been enlarged from a two-stamp to a 45-stamp mill (Figs. 3 and 4).



FIG. 3.—ABOVE THE YANGDEI MILL. TRIBUTE ORE IS SHOWN ON DUMP.



FIG. 4.—MILL AND CYANIDE PLANT OF THE CHIKSAN CO.

The course of the ore through the mill is shown in the flow sheet, Fig. 5. From the tippie bins the ore is drawn through chutes into 1¼-ton cars and trammed to the mill, where it is dumped over grizzlies, 3¼ ft. wide by 8 ft. long, the bars being set 1½ in. apart. The oversize goes to three Blake crushers of the following sizes: 9 by 9, 10 by 14, and

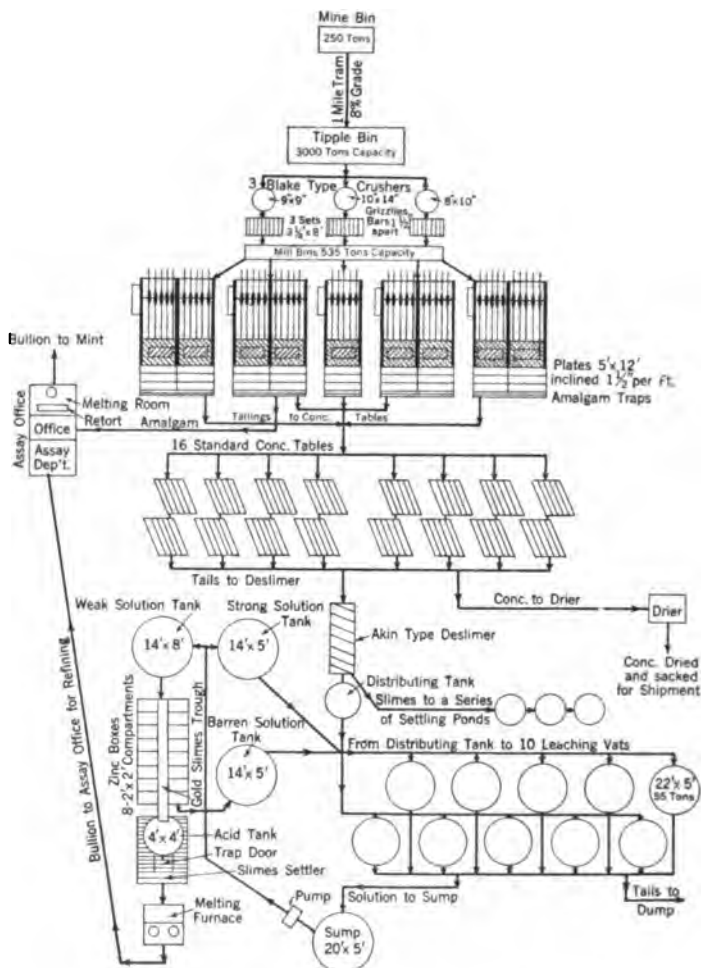


FIG. 5.—FLOW SHEET OF THE YANGDEI REDUCTION PLANT OF THE CHIKSAN CO.

8 by 10. After crushing, the ore drops direct to the mill bins. The ore then gravitates through nine Challenge feeders, which supply the nine batteries. The stamps weigh 1,250 lb. and are operated at 100 7-in. drops per minute. The distributed weight of the stamp is: Stem, 610 lb.; tappet, 150 lb.; boss head, 300 lb.; and shoe, 190 lb.

The ore is crushed to 30 mesh, and is then run over amalgamating

plates, both inside and outside amalgamation being in use. The plates are 5 by 12 ft. and have a slope of $1\frac{1}{2}$ in. to the foot. The pulp passes from the plates into Pierce amalgamating traps, and from the traps is distributed to the first series of eight concentrating tables. The middlings from these tables go to another series of eight tables on the floor below. The concentrates are dried, sacked, and shipped to the United States.

After the completion of the deslimmer, the tailings will be run direct to vats in the cyanide plant; at present they are stacked before cyanide treatment.

Cyanide Plant

In August, 1913, the cyanide plant commenced operations. At that time six vats, 22 ft. in diameter by 5 ft. deep, with a capacity of about 55 tons of damp sands, were used for leaching; during the year it was necessary to add four more vats of the same type to handle the mill tailing.

An Akins deslimmer is being installed and will be ready for operation early in 1915. The sand will then be run from the deslimmer through a rotating launder distributor, which will supply the vats; slime from the deslimmer will be run into a series of settling ponds, and if found to contain sufficient value, will be cyanided after a large enough amount has accumulated.

At present the vats are filled and emptied by coolies, carrying "jiggies." This is a very uncertain method, for when the planting and harvesting seasons come the coolies go to work on the farms, leaving the cyanide plant short handed. For this reason the plant can be run to its full capacity only a few months of the year.

The sand in the leaching vat is first treated with a 0.2 per cent. solution of potassium cyanide, supplied to the vat through a $2\frac{1}{2}$ -in. pipe from the solution tank. After about 7 hr. the solution is drained off, and while draining a 0.1 per cent. cyanide solution is run in; this is drained after a few hours, and at least two washings of water applied. The vats are then unloaded and prepared for a new charge, four days being taken for total treatment, including the loading and unloading of the vats.

The pregnant solution is pumped up to the receiving tank, from which it flows to two 8-compartment zinc boxes, having a total length of 20 ft., with compartments 2 ft. square. Zinc shavings are used as the precipitant. The solution from the boxes is run into the barren-solution tank, and is then pumped up to the strong-solution tank, where it is made up to the required strength.

The gold slime is cleaned from the boxes and run into the acid tank, where for three days it is treated with sulphuric acid. The slime is then run into a vacuum settler and drained; after which it is dried over a large iron drying pan, weighed, and melted, the resulting button being sent to the assay office for refining and assaying.

The performance of the mill for the year ended Dec. 31, 1914, was:

Tons crushed.....	46,402
Average assay value.....	\$6,956
Gross value (head assays).....	\$322,770
Bullion (21,872 oz.):	
Gold.....	\$231,932
Silver.....	5,794
Concentrates (414.33 tons).....	31,388
Total.....	\$269,114
Milling cost, per ton crushed.....	
Average assay value of concentrates.....	\$0.67
The stamp days.....	\$75.75
Stamp duty per 24 hr.....	11,190.24
	4.12 tons.

The costs and recovery at the cyanide plant for the year Jan. 1, 1914 to Jan. 1, 1915, were:

	Per Ton	Total
Recovery.....	\$0.799	\$29,426
Cyaniding costs.....	0.335	12,462
Profit.....	\$0.464	\$16,964

Total recovery from reduction plant was:

Bullion (28,105.6 oz.):	
Gold.....	\$259,013
Silver.....	8,062
Concentrate.....	31,388
Total output.....	\$298,463

Water for the mill is brought from the creek through a small ditch about a mile long. During the summer or dry season the water supply was insufficient to run the mill, so it was necessary to make settling ponds; the water from these ponds is pumped back to the mill.

The mill is operated by a 300-hp. Russell compound engine, which also drives a 75-kw. alternating-current generator, which in turn drives the other small 5- and 10-hp. motors and supplies the lighting service for the mine and camp. Power for the engine is supplied by three 150-hp. boilers.

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The Formation and Distribution of Residual Iron Ores

BY C. L. DAKE, * ROLLA, MO.

(San Francisco Meeting, September, 1915)

RESIDUAL deposits occur both as products of weathering and as products of hydrothermal decay.

PRODUCTS OF WEATHERING

That climatic conditions affect greatly both the rate and the results of weathering, is shown in temperate, polar, arid, and tropical regions, each of which will be considered separately.

Weathering under Temperate Climates

The decomposition of any rock is accomplished by the removal of the more soluble bases, and a concentration of the relatively insoluble oxides of iron, aluminum, and silicon. By reason of its somewhat greater solubility, silica is much more subject to removal than iron and alumina. The two important residual end products of weathering are sands and clays, the former consisting essentially of quartz, and the latter of kaolin and limonite.¹

When such deposits are formed *in situ*, the sands and clays are not separated, but remain as mixtures, which can be more or less perfectly separated by treatment with water. In such a separation, the ferric oxide or hydrate tends, by reason of its fine state of division, to remain with the clays and silts rather than the sands. While ferruginous sandstones are everywhere common, they are rarely rich enough in iron to become an ore without secondary concentration. Since the silica of these sandstones is completely crystalline quartz, it is much less liable to removal by solution than are the more soluble forms of partly amorphous chert and jasper, which have been removed on so large a

* Non-member.

¹ Merrill, G. P.: *Rocks, Rock-Weathering, and Soils*, 2d ed., p. 289 (1906).

scale from the Lake Superior iron formations.² Moreover, the iron cement of sandstones, which constitutes the chief iron content of these rocks, is often a result of later infiltration, and not of original sedimentation. Thus if clays rather than sands contain the iron of residual decay, we are led to consider the former more carefully with a view to determining, as far as possible, the behavior of the iron with respect to the other constituents.

Since sands and clays, together with their cemented equivalents, sandstones, quartzites, shales, and slates, already represent the ultimate products of weathering, and consist of the more insoluble rock elements, we do not look to them to furnish residual products by further decay. In exceptional cases, such decay seems to go on. D'Inwilliers³ accounts for certain iron ores in Pennsylvania by the decay of damourite slates; but in general, sandstones and slates are the ultimate products beyond which chemical decay, under normal weathering conditions, cannot go.

Limestones, on the other hand, while representing an ultimate product of decay, are easily soluble, and if they contain insoluble impurities, these will be concentrated by solution decay. Such concentration is widely observed everywhere. Where limestone contains iron as one of the important impurities, the residuary products will be rich in iron.

The processes of decay already described are also active on igneous rocks, and result in residual clays and sands. Both acid and basic rocks are attacked; but in general, the more basic the rock, the more rapid the decay, and the greater the percentage of iron in the resulting clay. This results from two causes. First, the SiO_2 in the more basic rocks is all combined, and when freed by the removal of the bases, is in the colloidal form, and more soluble than the quartz of an acidic rock; and for that reason, its removal in solution will result in a higher degree of concentration. Secondly, basic rocks are, as a rule, originally higher in iron than acidic rocks. Everywhere in temperate regions where the humidity is high, such decay goes on rapidly and to great depths. The region of the southern Appalachians is a good example, with its deep soils and their brilliant reds and yellows.⁴

Concerning the efficiency of weathering in increasing iron content, considerable data are available. Chamberlin⁵ gives four limestone-clays

² Van Hise, C. R., and Leith, C. K.: *Geology of the Lake Superior Region, Monograph LII, U. S. Geological Survey*, p. 537 (1911).

³ D'Inwilliers, E. V.: *Iron Ore Mines of the Great Valley, Annual Report, Pennsylvania Geological Survey*, 1886, pt. 4, p. 1411.

⁴ Russell, I. C.: *Subaerial Decay of Rocks and Origin of the Red Color of Certain Formations, Bulletin No. 52, U. S. Geological Survey*, p. 22 (1889).

⁵ Chamberlin, T. C., and Salisbury, R. D.: *Preliminary Paper on the Driftless Area of the Upper Mississippi Valley, Sixth Annual Report, U. S. Geological Survey*, p. 250 (1884-85).

from the Driftless Area of Wisconsin, containing respectively 5.52, 8.53, 10.04, and 17.19 per cent. of ferric oxide.⁶ Analyses of the parent limestone are not given, but Grant⁷ gives two analyses of typical limestones from the same area and formation, showing respectively 0.9 and 0.95 per cent. of Fe_2O_3 . The Karst limestone, with only 0.044 per cent. of ferruginous silicates, yields the famous Terra Rossa, with a marked iron content.⁸ Other cases of limestone weathering show concentration of iron as follows:

	Limestone Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Virginia ⁹	0.19	10.5
Knox dolomite ¹⁰	0.23	8.3
Lexington, Va. ¹⁰	$\left. \begin{array}{l} \text{Fe}_2\text{O}_3 \\ \text{Al}_2\text{O}_3 \end{array} \right\}$ 0.42	15.16
Bermuda (red earth soils) ¹⁰		
	$\left. \begin{array}{l} \text{Fe}_2\text{O}_3 \\ \text{Al}_2\text{O}_3 \end{array} \right\}$ 0.54	13.898

Weathering of acid igneous rocks shows concentration of iron as tabulated below:

	Unweathered Rock Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Micaceous gneiss ¹¹	9.06	12.18
Granite ¹²	4.00	4.39 (and FeO)
Biotite granite ¹³	1.45	2.44
Biotite granite ¹⁴	1.41	1.91
Biotite granite ¹⁵	1.96	6.33
Porphyritic biotite granite ¹⁶	2.51	3.45
Granite ¹⁷	3.46	5.04
Granite ¹⁸	1.44	2.05
Hornblende granite ¹⁸	3.43	2.67

Acid rocks in general show upon decay a concentration of iron which is slight, as compared with that shown by limestones.

⁶ For the sake of uniformity, the FeO is recomputed as Fe_2O_3 in this and all the following comparisons.

⁷ Grant, U. S.: Lead and Zinc Deposits of Wisconsin, *Bulletin No. 14, Wisconsin Geological and Natural History Survey*, pp. 13, 33 (1906).

⁸ Merrill, G. P.: *Rocks, Rock-Weathering, and Soils*, 2d ed., p. 375 (1906).

⁹ Watson, T. L.: Lead and Zinc Deposits of Virginia, *Bulletin No. 1, Virginia Geological Survey*, p. 98 (1905).

¹⁰ Russell, I. C.: *Bulletin No. 52, U. S. Geological Survey*, p. 24 (1889).

¹¹ Merrill, G. P.: Weathering of Micaceous Gneiss in Albermarle County, Virginia, *Bulletin of the Geological Society of America*, vol. viii, p. 157 (1896).

¹² Merrill, G. P.: Disintegration of the Granitic Rocks of the District of Columbia, *Bulletin of the Geological Society of America*, vol. vi, p. 321 (1894).

¹³ Watson, T. L.: Granites and Gneisses of Georgia, *Bulletin No. 9A, Georgia Geological Survey*, p. 302 (1902).

¹⁴ *Idem*, p. 312.

¹⁵ *Idem*, p. 315.

¹⁶ *Idem*, p. 321.

¹⁷ Holland, P., and Dickenson, E.: *Proceedings of the Liverpool Geological Society*, vol. vii, pt. 1, p. 116.

¹⁸ Culver, H. E.: The Formation of Laterites: A Thesis Submitted for the Degree of Master of Philosophy, at the University of Wisconsin, p. 39 (1911).

Below are tabulated examples of the concentration of iron shown by the weathering of basic igneous rocks.

	Fresh Rock Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Diorite ¹⁹		15.94
Greenstone ²⁰		18.00
Diorite ²¹		18.00
Diabase ²²	11.54	13.37
Diabase ²³	11.60	35.96
Diabase ²⁴	13.54	40.68

Residual decay in temperate regions yields sediments rich in iron as compared with the original rocks. Only limestones and basic igneous rocks, however, show sufficient concentration to produce deposits rich enough to approach ores.

Weathering in Polar Regions

Polar regions with their low temperatures are regions of disintegration rather than of decomposition.²⁵ Such chemical decay as has occurred seems to be of the same general type as that of the temperate regions, the presence of ocherous clays²⁶ indicating that some chemical decomposition goes on. Such decay, however, as far as observed, is not sufficient to produce any large bodies of iron-rich sediments.

Weathering in Arid Regions

In arid regions disintegration is also much more characteristic than decomposition.²⁷ In the valley of Lower California, sediments are forming which consist largely of undecomposed rock material. Walther concludes²⁸ that chemical weathering in the desert is of slight importance. Even where such weathering does go on in the desert, the soluble constituents tend to remain, because the water supply is too slight to

¹⁹ Ries, H.: Clays of Wisconsin, *Bulletin No. 15, Wisconsin Geological and Natural History Survey*, p. 117 (1906).

²⁰ *Idem*, p. 131.

²¹ *Idem*, p. 174.

²² Holland and Dickenson: *Proceedings of the Liverpool Geological Society*, vol. vii, p. 108.

²³ Watson, T. L.: Weathering of Diabase near Chatham, Virginia, *American Geologist*, vol. xxii, No. 2, p. 87 (Aug., 1898).

²⁴ Merrill, G. P.: Disintegration and Decomposition of Diabase at Medford, Massachusetts, *Bulletin of the Geological Society of America*, vol. vii, p. 350 (1895).

²⁵ Van Hise, C. R.: A Treatise on Metamorphism, *Monograph XLVII, U. S. Geological Survey*, p. 498 (1904).

²⁶ Walther, J.: *Einleitung in die Geologie*, vol. iii, p. 740.

²⁷ Van Hise, C. R.: *Monograph XLVII, U. S. Geological Survey*, p. 496 (1904).

²⁸ Walther, J.: *Op. cit.*, p. 777.

remove them.²⁹ If this is true, there would be little concentration of insoluble material, since this concentration is normally brought about by the removal of the soluble constituents.

Hilgard,³⁰ on the other hand, shows a distinctly higher percentage of iron in desert than in humid soils. He states, however, that there is no obvious chemical reason for this greater abundance, and assigns as a possible cause that the soils chosen to represent arid regions resulted in many cases from rocks originally high in iron.

Residual deposits of iron are probably not forming in desert regions in quantities worthy of consideration.

Tropical Weathering

In tropical regions, weathering is more intense than in temperate climates, and proceeds more rapidly and to greater depths. There is also, in certain regions, a difference in the residual products; instead of the characteristic clays of temperate regions, laterites often result. It seems to be well established, however, that true clays as well as laterites are formed in tropical regions.

Lateritic Deposits.—Since the term laterite has been so loosely used, it is necessary to adopt a definition as a basis for further discussion. Clarke³¹ defines laterite as “essentially a mixture of ferric hydroxide, aluminum hydroxide, and free silica in varying proportions.” Culver,³² who has made a close study of laterites, follows Clarke in his definition, and the term will be thus limited in this discussion. As defined above, laterites differ from true clays in the absence of the aluminum silicate, kaolin, and the presence in its place of some aluminum hydrate. Culver, in the work cited above, concludes, from the examination of all the available data, that laterites develop only in tropical and semi-tropical regions of heavy rainfall and only from basic rocks. The wide extent and great depth of lateritic deposits make their consideration of prime importance in the study of iron-rich residual deposits. Moreover, laterites represent a more complete decay of rocks and a greater removal of silica than do ordinary clays. Therefore they must show a proportionate increase in the remaining constituents, and consequently, other factors remaining constant, a residual deposit richer in iron. To illustrate, Culver³³ gives an example of a clay which contains 47 per cent.

²⁹ Van Hise, C. R.: *Loc. cit.*

³⁰ Hilgard, E. W.: *Soils*, p. 392.

³¹ Clarke, F. W.: *The Data of Geochemistry, Bulletin No. 491, U. S. Geological Survey*, p. 470 (1911).

³² Culver, H. E.: *The Formation of Laterites: A Thesis Submitted for the Degree of Master of Philosophy at the University of Wisconsin (1911).*

³³ *Idem*, p. 53.

SiO_2 , and 14.6 Fe_2O_3 . He also gives a list³⁴ of typical laterites showing the content of SiO_2 from 0.37 to 10.75, with an average of less than 3 per cent. If the SiO_2 were removed from the above clay to form a typical laterite, it would mean a loss of SiO_2 expressed by the following equation: $\frac{100-x}{100} \times 3 = 47 - x$,³⁵ from whence $x = 45.36$ per cent. of the original weight. The residual portion would then be represented by $100 - 45.36 = 54.65$ per cent. of the original weight. The iron content of the residual deposit would then be increased from 14.6 per cent. of the original weight to $\frac{100}{54.65}$ of 14.6, or 26.72 per cent. of the weight of the residual mass. This indicates clearly one reason for the high content of iron in laterites. Another reason is doubtless their formation from basic rocks only. The degree of concentration of iron shown in lateritic decay is expressed in the following table:

	Fresh Rock Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Epidiorite (British Guiana) ³⁶	4.66.....	$\left\{ \begin{array}{l} 25.60 \\ 26.73 \end{array} \right.$
Diabase (British Guiana) ³⁷	10.70.....	$\left\{ \begin{array}{l} 22.31 \\ 24.82 \end{array} \right.$
Dolerite (India) ³⁸	13.69.....	23.40

A series of laterites from India³⁹ run 13.75, 23.41, 26.61, 34.37, 37.88, 47.27, 51.25, and 56.01 per cent. Fe_2O_3 . The parent rock is not given. Two laterites from Surinam⁴⁰ contain 55.94, and 62.08 per cent. ferric oxide. Merrill⁴¹ cites an Indian laterite with 55.5 per cent. Fe_2O_3 . The parent rock is not given. Culver⁴² cites certain unpublished data furnished by Prof. W. J. Mead, of the University of Wisconsin, on the lateritic iron ores of Cuba. The original is a serpentine rock, the freshest of which contains 9.8 per cent. Fe_2O_3 . The residual lateritic ore, sampled every foot from surface to bed rock, 28 samples in all, averages

³⁴ Culver, H. E.: *Idem*, pp. 56 to 60.

³⁵ Let x = loss of SiO_2 in per cent. of original rock.

Then $47 - x$ = remaining SiO_2 in per cent. of original rock.

And $100 - x$ = total remaining material in per cent. of original rock.

But 3 = remaining SiO_2 in per cent. of total remainder.

And $\frac{100-x}{100} \times 3$ = remaining SiO_2 in per cent. of original rock.

Whence $\frac{100-x}{100} \times 3 = 47 - x$ and $x = 45.36$ per cent.

³⁶ Culver, H. E.: *Op. cit.*, p. 30.

³⁷ *Idem*, p. 43.

³⁸ *Idem*, p. 54.

³⁹ *Idem*, pp. 59 to 60.

⁴⁰ *Idem*, p. 62.

⁴¹ Merrill, G. P.: *Rocks, Rock-Weathering and Soils*, 2d ed., p. 299 (1906).

⁴² Culver, H. E.: *Op. cit.*, p. 45.

64.3 per cent. Fe_2O_3 . The data above show that lateritic decomposition is very efficient in the concentration of iron, in residual deposits.

Tropical Clays.—Culver, in the paper cited above, says, "No deposit of true laterite is known which has certainly been derived from acid igneous rocks." He quotes Harrison's extensive work on laterites as showing conclusively that, under conditions where true laterites are formed from basic rocks, acidic rocks alter to true clays, with the normal kaolin content—clays entirely comparable to those of temperate regions.

General Distribution of Weathering Products

It may be said in general that residual deposits are not formed in desert and frigid regions. Such deposits, on the other hand, occur in abundance in humid temperate and tropical regions. Only areas covered either by limestones or basic igneous rocks yield residual deposits high in iron. True laterites, which represent the residuary material richest in iron, are developed only over areas of basic rocks, in tropical regions of heavy rainfall.

PRODUCTS OF HYDROTHERMAL DECAY

Hydrothermal metamorphism is frequently so profound as to change materially the chemical and mineralogical composition of large areas of rocks. It is only rarely, however, that this process can actually be seen occurring at the present time, under conditions which permit the changes to be traced through their various steps. It is still more exceptional to find recognizable cases where this alteration has gone to so complete a stage that the end products consist of residual soils, essentially in the ultimate stage of decay. So far as the writer knows, Maxwell⁴³ is the only investigator who has made a careful study of this method of formation of residual soils.

Nature of Hydrothermal Decay in the Formation of Soil

The rocks with which Maxwell deals, in the investigation above referred to, are wholly of basaltic type. He divides them into non-hydrous lavas, hydrous lavas, and tufas. These divisions are based upon what he assumes to be the original or primary alteration effected upon the lavas by steam and heated waters at the time of their extrusion. Originally representing the same magmas, they now differ in the degree of hydration and oxidation they have undergone. The iron content of these lavas is as follows:

⁴³ Maxwell, Walter: *Lavas and Soils of the Hawaiian Islands: Rept. of the Experimental Station of the Hawaiian Sugar Planters' Association, Special Bulletin A, Department of Agriculture and Chemistry, Honolulu (1905).*

	FeO Per Cent.	Fe ₂ O ₃ Per Cent.	Total Iron Recomputed as Fe ₂ O ₃ Per Cent.
Non-hydrous lavas.....	8.43	4.04	13.41
Hydrous lavas.....	8.66	5.62	15.24
Tufas (Tuffs).....	1.69	25.79	27.67

The above are the averages of a large number of individual analyses of each type. Two important points are brought out in the above figures: First, there is a definite relative increase of the ferric over the ferrous oxide in the change from non-hydrous lavas to tuffs, and second, there is an apparent increase in the total iron content. This apparent increase is probably due to the loss of the soluble constituents, the silica decreasing from 49.12 to 26.60 and the lime from 9.24 to 1.41 per cent. It will at once be noted that the change is quite comparable to the lateritic process just discussed.

After the lavas have been ejected and solidified, there are large areas in the Hawaiian Islands where solfataric action still further modifies the composition of the rocks. These changes are brought about by the action of intensely acid steam, in which the sulphuric acid sometimes reaches 5 per cent. or more. From the floors of these solfataras, blocks of partly altered lava were taken, while yet hot and while being acted upon by the steam ascending through the fissures in the floor. These showed the lava in all stages of decay. A peculiarity of the outcome of the process is that all the products seem to be in a sense residual: gypsum, silica, alum, ocher. The products are, moreover, segregated in alternating layers, irregular nodules, vast pockets, and other complex relations. Large masses of white siliceous soils alternate with areas of red ocherous earths, and these in turn with gypsum. The essential point of his discussion, however, as applied to the theme of this paper, is that hot waters and gases, carrying abundant sulphuric acid, have been proved to effect the complete decomposition of basic lavas, and the segregation of the products of the decay, so that certain of the resultant soils show a very marked increase in the proportion of ferric oxide, as compared with that in the original rocks.

Efficiency of Hydrothermal Decay in Increasing Iron Content

It must be borne in mind, in discussing the increase in iron due to decay by hydrothermal agencies, that the residual deposits vary widely in character. The following is a list of the more common end products, with their iron content, all of which are said to exist in large quantities.

	Fe ₂ O ₃ Per Cent.
Gypsum soils.....	0.7
Alum soils.....	12.3
Impure siliceous residue.....	4.9
Ocherous deposits.....	44.5

In discussing those residual soils of Hawaii, which he believes to be the result of the action of steam and hot water, Maxwell classes them according to color. Eighteen analyses of white and gray soils show from 0.7 to 18.62 per cent. Fe_2O_3 , with an average of 6.01 per cent. The Fe_2O_3 content of nine brown and yellow soils varies from 3.25 to 26.68 per cent., and averages 10.82 per cent. Twenty analyses of red soils vary between 16.99 and 83.68 per cent. in Fe_2O_3 and average 41.37 per cent. These figures show beyond a doubt that the red soils, which are so abundant in the islands, are rich in iron. These soils among others are ascribed by Maxwell to the action of hot water and steam. In summing up, he gives the following figures, compiled from a large number of individual cases, as the iron content of the average lava and the average soil of the islands: lavas, 13.36 per cent.; and soils, 36.45 per cent., of FeO and Fe_2O_3 .

Over what areas hydrothermal alteration is now, or has been in the past, effective in the enrichment of iron sediments is a question not easy to answer. It seems entirely probable that in regions of intense effusive activity, especially where the basic and more easily decayed types of lava are found, conditions might prevail entirely similar to those found by Maxwell in the Hawaiian Islands. Further investigation along this line would be very desirable.

It is quite possible that the decay of the Deccan traps⁴⁴ and the basalts of Antrim,⁴⁵ resulting in their accompanying iron-rich deposits, as well as other cases of a similar nature, may have been due in part, at least, to thermal alteration at or shortly after extrusion. The scale on which this alteration has taken place in the Hawaiian Islands leads us to believe that hydrothermal decay might be an important contributory factor in the formation of iron-rich residual deposits.

ALTERATION IN SITU OF PREVIOUSLY FORMED IRON DEPOSITS

The oxidation of cherty iron carbonate, and the subsequent leaching of the silica, with resultant enrichment of the iron formation to form ore,⁴⁶ as well as the weathering of pyrites to form gossan orebodies,⁴⁷ is a type of weathering resulting in what may be called residual deposits. But since this paper deals only with the primary accumulation of iron formations, such local secondary enrichment may be dismissed without further mention here.

⁴⁴ Foote: *Memoirs of the Geological Survey of India*, vol. xii, p. 200.

⁴⁵ Geikie, Sir A.: *Ancient Volcanoes of Great Britain*, vol. ii, p. 204.

⁴⁶ Van Hise, C. R., and Leith, C. K.: *The Geology of the Lake Superior Region, Monograph LII, U. S. Geological Survey*, p. 529 (1911).

⁴⁷ Moxham, E. C.: *The Great Gossan Lead of Virginia, Trans.*, xxi, 133 (1892-93).

SUMMARY

Residual soils rich in iron are produced both by weathering and by hydrothermal alteration.

Weathering produces important accumulations of iron sediments only from limestones and basic igneous rocks. In other rocks, such concentration is of minor importance.

Only temperate and tropical regions of moderate to heavy rainfall yield residual weathering products in abundance. Polar and arid regions are more favorable to mechanical disintegration.

Laterites, which by reason of their lower silica content and the nature of the parent rock are richer in iron than ordinary residual clays, are developed only from basic igneous rocks, and in tropical regions of heavy rainfall. Such laterites are widespread in the tropics, and represent enormous concentration of residual iron.

Hydrothermal decay has, in the Hawaiian Islands, produced iron-rich residual soils very comparable to laterites. Observation is limited, and we know little of the distribution or importance of this type of residual deposits.

Of the above types, hydrothermal decay, normal weathering, and lateritic weathering, the latter is probably of the greatest importance. This is due both to the great abundance and to the higher iron content of the laterites.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Method of Making Mineralogical Analysis of Sand*

BY C. W. TOMLINSON, † MINNEAPOLIS, MINN.

(San Francisco Meeting, September, 1915)

INTRODUCTORY

THE analyses which have been made by the writer according to the method described below were made as part of Professor Withey's investigation of the concrete aggregates¹ of Wisconsin, in the hope of throwing light on the extent and nature of the influence of the mineralogical constitution of aggregates upon the properties of the concrete and mortar made from them.

This method was found most satisfactory after four weeks of experiment and modification. It attains an accuracy quite sufficient for the purpose of the analysis, and is fairly rapid. Mineralogical analyses may be thus made at a cost considerably less than that of chemical analyses of a similar degree of completeness.

The results obtained by this method should not vary in any item by more than 1 per cent. of the total weight of the sample, and may be made very much better than this by the exercise of greater care in the microscopic counting (see below). The accuracy of the hand sorting is of course dependent upon the experience and carefulness of the analyst.

After a month's practice, the writer was enabled to turn out these analyses at the rate of four in 33 hr. of working time. It was found most economical to handle four samples at once.

The samples analyzed weighed from 290 to 340 g. each. The same method could be applied to larger or to smaller samples, within reasonable limits.

The following list covers the necessary apparatus and equipment:

- 1 good balance capable of weighing accurately to tenths (or, better, hundredths) of a gram, and of handling quantities up to 200 g. in weight.
- 1 Westphal balance, registering densities to two or three decimal places.
- 1 Meker or Bunsen burner, and gas supply.
- 1 nest of sieves, 10, 20, 40, and 100 meshes to the inch.

* Developed at the University of Wisconsin in June and July, 1914, by C. W. Tomlinson, under the direction of M. O. Withey and A. N. Winchell.

† Non-member.

¹ The *aggregate* comprises all of the material entering into the composition of a block of concrete, other than cement and reinforcement. Gravel, sand, crushed stone, and slag often constitute concrete aggregates.

- 1 binocular or petrographic microscope (not necessary for rough work).
 1 good-sized hand lens of moderate power.
 1 rubber laboratory apron.
 1 pair rubber gloves.
 Chemicals: potassium iodide and mercuric iodide (see following pages).
 Bottles to hold heavy solution.
 1 1,000-cc. wash-bottle.
 6 Harada tubes, if these are used.
 Beakers, as follows: two 500-cc.; three 200-cc.; five 100-cc.; three 50-cc.
 Casseroles: two 4-in.; two 2-in.; one 6-in.
 1 12-in. evaporating dish.
 4 4-in. glass funnels, for filtering.
 3 8-in. glass rods, $\frac{3}{16}$ in. diameter, for stirring.
 8-in. and 6-in. filter paper.
 5 yd. toweling.
 1 tripod support for evaporating dish.
 2 ring stands with 2 rings each, and clamps for supporting separation beakers.
 1 5-in. square of wire screening.
 1 counting sheet (see paragraph on "Microscopic Analysis").
 2 by 3 in. and 3 by 4 in. envelopes for filing separated sands.
 (Envelopes used for steel pens are suitable).
 Larger envelopes for filing separates of each sample.

OUTLINE OF METHOD

The various steps in the process of a complete analysis of an average or complex sand, with the time (figured for four analyses conducted at once) required by each, are as follows:

	Hours	Minutes
Sizing.....	1	15
Weighing of sizes.....	..	45
Preparation of heavy solution.....	2	..
Heavy-solution separations.....	5	..
Hand sorting.....	12	20
Weighing of separates.....	3	..
Microscopic analysis of heavy solution of separates.	3	30
Computation of analysis and preparation of report.	4	30
Total for four analyses.....	32	20
Time charged to each analysis.....	8	5

The chief saving in time from carrying on several analyses at once is in the heavy-solution work. The heavy-solution separations and the hand sorting may be carried on together, the latter filling the intervals during which the solution is being evaporated.

Each of the several steps is described in detail below.

Sizing

A separation of the sample into several sizes is advisable for three reasons: (1) The coarser material can best be classified by hand sorting;

(2) great range in size of grain reduces the accuracy of the heavy-solution separations; and (3) the resulting information is of value, since the composition of the sand differs in the several sizes according to definite principles. Material too coarse to pass a 20-mesh screen is apt to contain many fragments of phaneritic igneous rock not yet disintegrated into constituent minerals, and therefore of variable specific gravity. Moreover, material of this size can be rapidly sorted by hand, and can be more satisfactorily and completely classified than by specific gravity alone (heavy-solution separation); and the separates can then be *weighed*.

The sizing is conveniently done with a nest of sieves of 10, 20, 40, and 100 mesh. For accurate work each sieve should later be handled separately to perfect the sizing.

Weighing of Sizes

This is done for two reasons: To serve as a check upon the possible losses in manipulation and upon possible errors in the final weighing of separates, and to determine the care with which each size shall be sorted. Following the weighing, the percentage of each size in each sample is computed. If any group amounts to less than 1 per cent. of the whole, it may be neglected in the analysis. Each size is laid out upon a sheet of paper or (better) inserted in an envelope of appropriate size, and labeled with the number of the sample. If a standard set of sieves is used, it is not necessary to write down the size, as this is recognized at a glance—unless envelopes are used. Envelopes should be free from leakage, and should be sealed by folding the open end or side, not by pasting down the flap.

Preparation of Heavy Solution

An excellent classification of the sand which passes the 20-mesh sieve may be made by means of heavy solutions. A solution of density 2.9 will separate practically all of the dark silicate minerals from the others. One of 2.7 will separate quartz (2.65) from calcite (2.72) and dolomite (2.8 to 2.9). One of 2.585 or 2.59 will separate orthoclase from quartz and plagioclase. The first two of these separations are highly satisfactory, but the third (density = 2.585) is less so because of the wide range in density of chert and quartzite due to varying porosity. Most chert will float with the orthoclase. Most common igneous rocks will fall with the carbonates in the group between 2.7 and 2.9; but these are seldom abundant in the sizes of sand submitted to heavy-solution separation. Graywackes behave like the igneous rocks. Mica is not to be trusted, because its great surface area interferes with proper separation; the heavy solution has a high surface tension.

Probably the most satisfactory solution is that of Sonstadt and Thou-

let, which consists of potassium mercuric iodide in water. Its preparation, characteristics, and manner of use are well described in Johannsen's *Manual of Petrographic Methods*, pp. 519 to 521. It is not necessary to evaporate on a water bath, as there prescribed, if the analyst will keep fairly close watch of the solution as it nears its maximum density. Evaporation in a dish resting directly on a wire screen on a ring stand over a burner is much more rapid and produces exactly the same result. Allow the solution to cool before filtering.

The quantity of solution needed for handling four analyses at once—to conduct eight or ten separations simultaneously—is about 2,000 g. This requires about 1,080 g. of mercuric iodide and 920 g. of potassium iodide, total cost about \$12.

Too much emphasis cannot be laid upon the necessity for caution in handling the Sonstadt solution. A few drops on the fingers occasionally during one day may not produce any noticeable effect at the time, but is sufficient to cause a swelling of the flesh which may stop the work for one or two days. The fingers become distended and rigid, and the skin discolored and sore, sometimes developing a rash. A drop of Sonstadt solution under the finger nail will bring pain quite readily. Elsewhere, except on a cut or bruise, it produces no immediate effect, and is therefore so much the more dangerous. Rubber gloves should be worn during all operations involving this solution; and new gloves should be obtained as soon as any hole appears in the old ones.

A stain of Sonstadt solution on the clothing may be readily removed in water if applied at once, or in alcohol later; but a rubber apron is an excellent preventive of such stains.

When the required amount of solution has been prepared, evaporated to a high density (anything over 2.9 is sufficient), cooled, and filtered (filtering may be omitted at the judgment of the analyst), it should be brought to the density wanted for the first separation, by diluting with water. This is best done by diluting a part of it to a lower density than that desired, and then pouring the *heavier* fluid *into the lighter*; the lighter is placed in a beaker inclosing the pendant of a Westphal balance, and the heavier poured in from a casserole or other vessel with a good lip. As the proper density is approached, the introduction of the heavier fluid may be made drop by drop, and the desired density thus obtained with an accuracy of three or even four decimal places.

Heavy-Solution Separations

A full description of the nature and use of the various types of apparatus used for heavy-solution separations is to be found in Johannsen's *Manual of Petrographic Methods*, pp. 547 to 556. The simple Harada tube described on p. 549 is very convenient, and a single separation there-

with is quite satisfactory for the purposes of this work, without the repetition prescribed for more accurate results. An ordinary beaker may be used with practically as good results, however, according to Goldschmidt's method, described on pp. 548 to 549 (Johannsen); this has the advantages of cheapness of apparatus and wide range in size of beakers (to accommodate different quantities of sand); and, what is an even greater convenience, the beakers can readily be wiped dry.

Before pouring heavy solution, the density of which has been accurately adjusted to a desired figure, into a separation beaker, it is important that the interior of the beaker should be quite dry; for a few drops of water may lower the density of the fluid enough to impair the results of the separation seriously.

In making each separation, the beaker should be chosen of such a size as to accommodate the parcel of sand with about twice an equal volume of solution, to insure a clean separation in decanting the two groups of minerals. If the quantity of sand in one size is too great to be satisfactorily handled, it may be reduced by quartering; and the proportions of minerals found in the tested portion are then assumed to hold for the remainder also.

It is a waste of time to drain off the heavy solution from the sands by filtering, after the separation is completed. The two separate parcels of sand (one of less and one of greater density than the heavy solution) should be washed directly into two separate casseroles, and there thoroughly washed by repeated decantations. A 1,000-cc. wash bottle is of convenient size for this work: a smaller one requires refilling too often. All of the solution used in the separation goes into the wash water, and is subsequently evaporated to high density again, before the next separation is attempted. During the process of evaporation the analyst may busy himself with the hand sorting of the coarser sizes of the samples with which he is working; though he should keep fairly careful watch of the progress of the evaporation.

As already remarked, the writer has found it economical to handle four or more samples at once. If four are taken, and there are two sizes from each sample to be submitted to classification by heavy solution (it is often the case that the "finer than 100-mesh" group is too small in quantity to warrant analysis), there will be eight parcels to be so treated, four of size 20 to 40 (passed 20-mesh, but not 40-mesh screen) and four of size 40 to 100. If the "finer than 100-mesh" group is to be treated in each case also, there will be 12. The requisite number of beakers (common or Harada) should be selected, of sizes appropriate to the sizes of the parcels of sand; and an adequate quantity of heavy solution should be prepared of the desired density. A 500-cc. beaker will hold enough (about 1,400 g.) for six or eight moderately large separations (50 to 75 g. of sand in each). One important economy of time effected by large-

scale operation is in the matter of preparation and adjustment of the heavy solution.

It is usually best to perform the separation at density 2.7 first. Nearly all sands contain some material on each side of this mark, and it is almost always the most effective separation. If either of the separates, the heavy or the light, in any sample then is very small in quantity, no further separation need be made of it; otherwise, the eight (or twelve) parcels of sand *heavier than 2.7* will be treated to another separation at 2.9, and those *lighter than 2.7* to one at 2.585.

For the same reason that the separation beakers should be dry before the Sonstadt solution is inserted, the sands should be thoroughly dry before a second separation is attempted. For this purpose each parcel, as soon as it has been cleansed of heavy solution, is washed from its casserole on to a pad of filter paper (three or four sheets in thickness) labeled with the number of the sample. The size and density of each parcel can be recognized at a glance, after a little practice. These may be left in the sun, or laid out on the table in the vicinity of the burner used in evaporating the wash water. If the latter method is used, the sands must be allowed to cool when dry before being subjected to another separation; as any notable rise of temperature may change the density of the heavy solution.

As in the first preparation of the heavy solution, the concentrate from the wash water of the first separation should first be allowed to cool, then filtered, then brought to the exact density wanted for the second or third separation. A 12-in. evaporating dish, a ring stand, and a Meker burner make an excellent combination for the evaporation.

Three separations of two sizes from four samples result in four separates from each size in each sample, or a total of 32 separates. When all are dry, they should be placed in separate envelopes (free from leakage), each appropriately labeled, as for example:

Sd. 8 20-40 D 2.585-2.7

Hand Sorting

The materials in each sample which do not pass the 20-mesh screen are again separated for convenience by a 10-mesh screen. If the quantity of plus 10-mesh gravel is less than 50 g., or that of 10-to-20-mesh sand less than 10 g., the entire parcel may be sorted by hand. For larger quantities it is desirable to quarter down the parcel until a representative portion of reasonable size is obtained. It is somewhat risky, however, to sort much less than 10 per cent. of a parcel and to assume that the composition of the sorted portion then represents accurately that of the whole. The time required for hand sorting depends wholly upon the extent to which this assumption is carried, and the ease and rapidity

with which the analyst can recognize and classify the grains. One little trick which the writer has found to save time is this: Do not pick up each grain with forceps and drop it in the appropriate pile, but separate one kind of mineral or rock from the entire parcel (or part of it which is to be sorted) first, then another, etc.; simply using a lateral motion *on the paper*, sweeping the grains of one kind one way, all others the other way, with one arm of the forceps or with some similar fairly dull-pointed instrument. The most abundant or the most easily recognized mineral or rock type should be separated first; then the separation of each successive type may be made in much less time than the one preceding.

The chief groups into which most Wisconsin sands fall are the following:

Translucent quartz.	Dolomite (and limestone).
Chert and feldspar.	Shales.
Quartzite, graywacke.	Granites (or all phanerites together).
Heavy minerals.	Greenstones (and slates).

Some of these are missing in many sands, and in some sands it may be found desirable to separate these still further. It is unfortunate that the separation of chert from feldspar is often difficult without the use of polarized light, as the two have very different origins. The source of the sand and the abundance of phanerites in it often show whether or not any considerable quantity of feldspar is to be expected in it.

The properties which are most reliable in a swift classification of sand grains are the following: diaphaneity, luster, form and cleavage, hardness, granularity, and color. The last-named property must be used with caution.

When the sorting is complete, each type should be put in a separate envelope, appropriately labeled, as for example:

Sd. 8 10-20 Tr. Qz.

Weighing of Separates

When the hand sorting and the heavy-solution separations are all completed, the entire batch of separates must be weighed, together with all of the unsorted portions of the various sizes, if such there be. The weight of each separate is written upon the envelope containing it. These weights are then recorded for each sample upon a preliminary analysis sheet, and each weight is computed into a percentage of the total weight of the sample. If any portion was not sorted, the percentages of the separates for that size are computed as if all had been sorted. Any appreciable errors found in the sums of weights for the sizes as

compared with those obtained before should first be looked into. In the case of the heavy-solution separates, some slight loss in manipulation is to be expected, but it should be kept under 1 per cent.

If the sum of all the heavy-solution separates of a certain density is only about 1 per cent. (or less) of the total weight of the sample, as is sometimes the case with the group of density greater than 2.9, it is not worth while to make a microscopic analysis of these separates, but they may be grouped together in the statement of analysis.

Microscopic Analysis of Heavy-Solution Separates

Since two or more minerals or rocks of different character may have the same density, and since the heavy-solution separation is subject to minor imperfections, as noted above, it is necessary to make an examination of each of the heavy-solution separates to determine its composition. A good hand lens is sufficient for this purpose, and a black sheet with a square pattern of white lines on it makes the most satisfactory background for a quantitative determination. A bacteriological counter is excellent. A representative portion of the separate is spread out evenly on a glass plate lying upon the counting sheet. If there are grains of one or several varieties scattered through a preponderance of one other mineral, as is usually the case, the former may be counted over a large area; the latter over a few of the squares included in that area, and the number multiplied by the proper factor to obtain the total. The proportions of the several minerals are reckoned to percentages, and recorded on a second preliminary analysis sheet. If the separate to be examined is a large one, more than one part of it may be subjected to counting, and an average taken. For sand of the 20-to-40-mesh type, 4 or 5 sq. cm. is a suitable area to cover for the count; for 40-to-100-mesh, 1 or 2 sq. cm. may be sufficient.

The weight of each separate having been previously computed into percentage of the total weight of the sample, the amount of each mineral (or rock type) present in the separate may be reckoned into comparable figures by means of the proportions found in the microscopic count. If the same mineral is present in more than one of the heavy-solution separates from one size of a given sample (due to inconstant specific gravity or to imperfect manipulation), the sum of these figures for that mineral in the several separates is the amount of that mineral in the size (still expressed in percentage of the total weight of the sample).

Computation of Analysis and Preparation of Report

The time included under this heading in the outline given above covers also other computations made during the progress of the analysis.

The amounts (in percentage of the total, not in grams) of each mineral and rock type found in each size are now entered on one sheet, and added to give the total amount for each mineral and rock. The analysis may then be summarized as follows, though other groups may be used if appropriate:

Igneous rocks.

Shale group.

Quartz group (includes quartz, chert, and quartzite).

Dolomite group (calcareous sediments).

Feldspar.

Heavy minerals.

Not examined (often the sand which passes 100-mesh—when less than 1.5 per cent. of the total).

The following is an example of the record of an analysis of the simplest type.

SAND 21

Mineralogical Analysis

Weighing of Sizes	Weight in Grams	Per Cent. of Sample
Plus 10 mesh.....	41.22	13.10
10 to 20 mesh.....	17.40	5.51
20 to 40 mesh.....	150.38	47.83
40 to 100 mesh.....	103.17	32.75
Minus 100 mesh.....	2.54	0.81
	314.71	100.00

Hand-Sorted Separates	Group I Coarser than 10 Mesh			Group II 10 to 20 Mesh		
	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample
Igneous rocks.....	6.47	15.7	2.05	0.53	8.7	0.48
Shale group.....	4.58	11.1	1.46	1.52	24.8	1.37
Quartz group.....	10.38	25.1	3.29	2.42	39.5	2.18
Dolomite group.....	19.79	48.1	6.30	1.14	18.6	1.02
Feldspar.....				0.45	7.3	0.40
Heavy minerals.....				0.07	1.1	0.06
Total sorted.....	41.22			6.13		
Not sorted.....				11.27		
Total of group.....	41.22	100.0	13.10	17.40	100.0	5.51

Heavy-Solution Separates	Group III 20 to 40 Mesh			Group IV 40 to 100 Mesh		
	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample	Weight in Grams	Per Cent. of Group	Per Cent. of Total Sample
D > 2.9	2.26	1.5	0.72	1.03	1.0	0.33
D 2.7 to 2.9	28.74	19.1	9.13	5.36	5.2	1.70
D 2.585 to 2.7	114.12	75.9	36.31	94.10	91.2	29.87
D < 2.585	5.26	3.5	1.67	2.68	2.6	0.85
	150.38	100.0	47.83	103.17	100.0	32.75

Microscopic Counts

	20 to 40 Mesh		40 to 100 Mesh	
	Per Cent. of Class	Per Cent. of Total Sample	Per Cent. of Class	Per Cent. of Total Sample
D 2.7 to 2.9:				
Dolomite group	93.2	8.51	98.8	1.68
Igneous rocks	5.7	0.52		
Quartz group	1.1	0.10	1.2	0.02
	100.0	9.13	100.0	1.70
D 2.585 to 2.7:				
Quartz group	99.2	36.02	96.9	28.94
Feldspar	0.8	0.29	3.1	0.93
	100.0	36.31	100.0	29.87
D < 2.585:				
Quartz group	14.2	0.24	17.5	0.15
Feldspar	85.8	1.43	82.5	0.70
	100.0	1.67	100.0	0.85

D > 2.9: Assumed to be all heavy minerals.

Summary

Igneous rocks	3.05
Shale group	2.83
Quartz group	70.94
Dolomite group	17.51
Feldspar	3.75
Heavy minerals	1.11
Not examined	0.81
Total	100.00

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Geology of the Burro Mountains Copper District, New Mexico

BY R. E. SOMERS, ITHACA, N. Y.

(San Francisco meeting, September, 1915)

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I. INTRODUCTION

1. Location, Topography, and Climate

The Burro Mountains are located in the southwestern part of New Mexico, in Grant County. The group is made up of two distinct mountain masses, known as the Little Burros and the Big Burros, the former lying about 6 miles southwest of Silver City, and the latter about 9 miles farther in the same direction.

They are separated by the valley of the Mangas River, a tributary of the Gila, and a stream only in name, except after a heavy downpour of rain. The Little Burros attain an elevation of 6,625 ft. above sea level, and the Big Burros 8,054 ft., with the surrounding country averaging about 6,000 ft. To the south of these mountains stretches the desert plain, while to the north are the more frequent elevations of the mountainous country.



FIG. 1.—BIG BURRO MOUNTAINS FROM LEOPOLD, N. M. SHOWS A PORTION OF THE DISSECTED SLOPE WHICH EXTENDS FROM THE BIG BURROS TO THE MANGAS VALLEY.



FIG. 2.—MANGAS VALLEY, SHOWING RIDGES OF PLEISTOCENE AND RECENT GRAVELS IN THE FOREGROUND.

The Little Burros are about 8 miles long, and $\frac{1}{2}$ to $1\frac{1}{2}$ miles wide, with a northwest-southeast trend. To the southwest they drop off rather abruptly into the Mangas Valley, but to the northeast their slope is gradual.—The continental divide passes through the southeasterly end of this group, as well as through the summit of the Big Burros.

The Big Burros are more rounded in plan. They descend rather rapidly on the north and east from their maximum elevation of 8,054 ft. above sea level, down to about 7,000 ft. altitude, from which there is a rather gentle slope for about 5 miles until the surface drops off into the Mangas Valley on the northeast. This slope is made up of ridges separated by sharp gulches, or *arroyos* (Fig. 1), which are tributary to the Mangas. Near the foot of this slope is situated the camp of Tyrone, with Leopold about 2 miles farther up in a southwesterly direction.

The climate is arid, and hence the ridges are barren except for the sage brush, cactus, and mesquite, which detract little from the yellowish brown, or, in the mineralized section, rusty and jagged, appearance of the surface. In some of the gulches, however, a few evergreens and other trees may be found, especially about Tyrone, where in places the desert-like appearance is wholly lacking.

The accompanying map (Fig. 3) shows the relative location of the different points of interest in the district.

2. *Scope of Work and Acknowledgments*

The Burro Mountains, as a whole, contain two classes of ore deposits: (1) quartz veins with gold and silver; and (2) copper deposits in fracture zones with more or less disseminated replacement of the walls. The former have not been very extensively worked, but the latter have attracted considerable attention, and the richest of them, so far as shown by present development, lie in the vicinity of the two camps, Tyrone and Leopold. The object of this paper is to set forth the results of a rather detailed study upon this section of the Burro Mountains area.

The field work was done during the summer of 1913, and was made possible through the kindness of Dr. James Douglas and Walter Douglas of the Phelps-Dodge Co. To E. M. Sawyer, Superintendent of the Phelps-Dodge properties at the Burro Mountains, and to his staff, the writer wishes to extend his thanks for their hospitality and the facilities which they so willingly placed at his disposal. To Prof. Heinrich Ries the writer is particularly indebted for generous and thoughtful guidance and patient criticism throughout the work.

3. *History and Mining*

Copper was first discovered in 1871, and in the following years the district passed through various periods of activity and idleness



FIG. 3.—MAP OF THE BURRO MOUNTAINS REGION, NEW MEXICO.

until, in 1905, the Phelps-Dodge Co. secured the Burro Mountain Copper Co., which was organized in 1904 by the Leopold Brothers, of Chicago, to work a group of claims around the camp which is now called Leopold. In the vicinity of Tyrone there was another group of claims which was operated by the Chemung Copper Co., and in 1913 the Phelps-Dodge Co. purchased these also.¹

Since 1904, development in the Burro Mountains has been going on steadily and with excellent results. Tyrone is being made the



FIG. 4.—SAMPSON SHAFT AT LEOPOLD.

center of activity and it is reached by the recently completed Burro Mountains R. R., which runs from Whitewater on the Deming to Silver City spur of the Santa Fé.

At Leopold the principal shaft is the Sampson (Fig. 4), and through it a number of orebodies, such as the East and West Sampson, and the Protection, have been worked in the past. A short distance northeast of the Sampson shaft is a large deposit of developed ore, called the East orebody. There is also at Leopold the St. Louis shaft, an incline which is on the so-called "St. Louis slip," or fissure. South of the St. Louis is the McKinley shaft, which, however, has not been extensively worked. About $\frac{1}{2}$ mile northeast of Leopold are the Boston and Niagara shafts,

¹ For more details, see Wade, W. Rogers: Burro Mountain Copper District, *Engineering and Mining Journal*, vol. xcvi, No. 7, pp. 287 to 289 (Aug. 15, 1914); also Bush, F. V.: Burro Mountain Porphyry Copper Developments, *Mining Press*, vol. cx, No. 6, pp. 222 to 224 (Feb. 6, 1915).

neither of which has shown very much ore. The former connects with the Sampson workings. At Tyrone there are three shafts, called No. 1, No. 2, and No. 3. The first, which is separate from the other two, lies to the northwest and is not yet very important. Nos. 2 and 3 (Fig. 5) lie close together in the same gulch in which Tyrone is situated and open extensive deposits. Between the 300 and 400 ft. levels in No. 3 is a rich deposit in breccia, which is known as the "breccia orebody." Between No. 2 and No. 3 is the portal of the Niagara tunnel, a 7,000-ft. opening, running southwest, and extending to the East orebody at Leopold. It is connected by a crosscut with the Niagara shaft, and is



FIG. 5.—SHAFT NO. 2. ENTRANCE TO NIAGARA TUNNEL IS NEAR HOUSE IN LOWER RIGHT-HAND CORNER.

to be used to deliver the ore from the Leopold workings to the railroad, which is to enter it on grade. About 1,400 ft. from the portal, the Niagara tunnel has opened up a mass of ore called the "Bison orebody."

Besides the larger interests mentioned above, the Savannah Copper Co., Mangas Development Co., and others, are doing some work in the district.

II. GENERAL GEOLOGY

1. *Introduction*

The Big Burros, considered broadly, consist of a pre-Cambrian granite complex into which has been intruded a mass of quartz monzonite of post-Cretaceous age. The summit of the Big Burros is of

granite, and the flanks are either granite or monzonite down to the level where both are covered by the gravels of Pleistocene and Recent age. These gravels represent the débris worn from the earlier rocks and washed into the valleys or out on to the plain.²

The vicinity of Tyrone and Leopold includes a corner of this monzonite mass, and some of the granite into which it has been intruded. In detail, the rocks may be enumerated as follows: (1) pre-Cambrian granites; (2) quartz porphyry, of uncertain age; (3) post-Cretaceous quartz monzonites; (4) volcanic breccia of Tertiary age; (5) Pleistocene and Recent gravels.

2. Granite

Megascopic Characters.—The granites are medium- to coarse-grained rocks, containing rather large grains of feldspar when fresh, and abundant irregularly shaped grains of quartz. Biotite is occasionally seen but it is not common. The feldspars are of two kinds, a pink orthoclase, and a white plagioclase, and their colors may be easily distinguished in the hand specimen when the rock is not badly altered. Sometimes the feldspars are enough larger than the other minerals to give a slightly porphyritic texture.

The granite, in whatever condition it may be found, can be recognized megascopically by its abundant, irregularly shaped quartz grains, and by the lack of small feldspar phenocrysts. The first feature distinguishes it from the quartz porphyries, in that their quartz is in the form of rounded hexagonal phenocrysts and is not as abundant as in the granite; and also from the monzonite, which seldom contains quartz in large enough grains to be visible in the hand specimen. The granite is not sufficiently silicified to furnish any obstacle to this method of recognizing it. The lack of small feldspar phenocrysts in the granite distinguishes it from the porphyritic varieties of the monzonite as well as the quartz porphyry. The granite is never porphyritic with the exception of the very coarse phase already mentioned.

Microscopic Characters.—Under the microscope, the plagioclase proves to be an albite or oligoclase (Fig. 7). The orthoclase usually shows an exceedingly fine perthitic intergrowth with a plagioclase, which is too minute to be identified. In the granite as a whole, plagioclase is usually more abundant than orthoclase. Zircon, titanite, and apatite are present as accessories.

Structure, Distribution, and Age.—The granite is the original country rock of the district. It is found chiefly in the northern part of the area studied (see Fig. 6), where, however, it contains numerous and

² For a very good description of the Burro Mountains region, see Paige, Sidney: *Ore Deposits near the Burro Mountains, New Mexico, Bulletin No. 470, U. S. Geological Survey*, pp. 131 to 150 (1911).

irregular dikes of monzonite, which gradually diminish in size and number beyond the lateral limits of the map. These dikes are so irregular that they were not mapped in detail, but they are so numerous that this section of the area cannot be called granite without qualification.

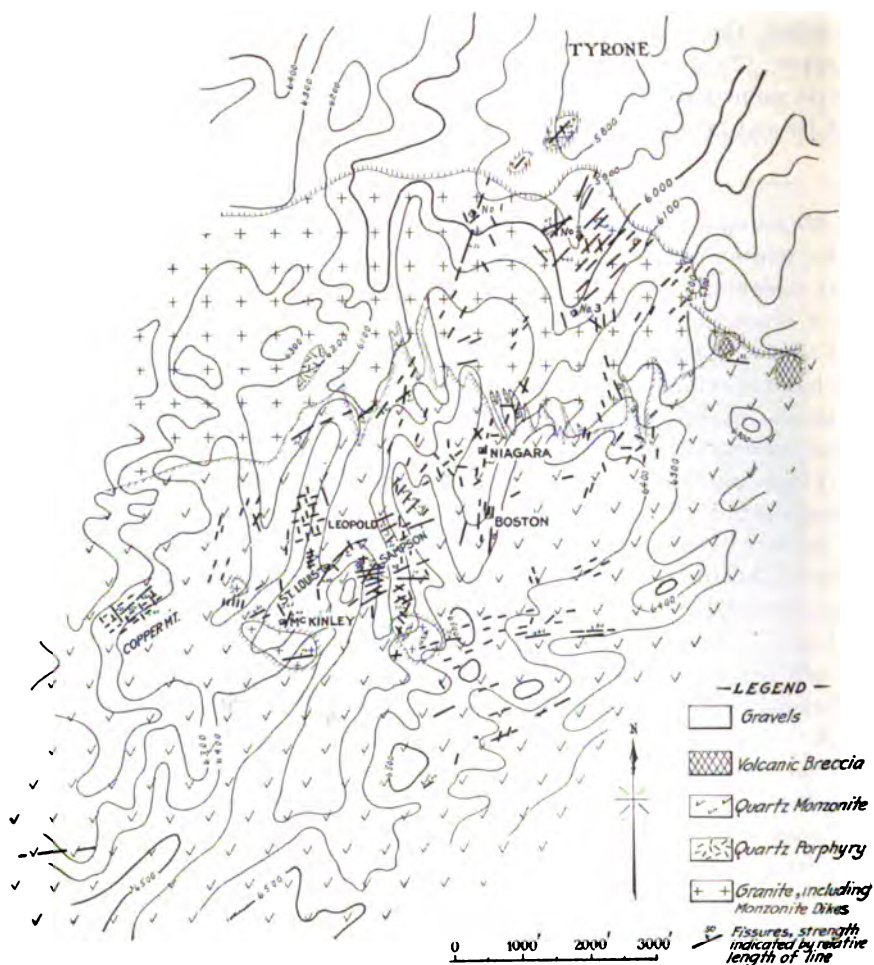


FIG. 6.—GEOLOGICAL MAP OF THE BURRO MOUNTAINS COPPER DISTRICT.

The granite included in this study of the Tyrone-Leopold district is but a very small part of the main mass. It is covered on the north by the Pleistocene and Recent gravels which fill the Mangas Valley, and does not appear directly opposite on the other side of the Mangas because of an intrusion of rhyolite porphyry which has cut through at that

point. It is found, however, about 4 miles north-northwest in the Little Burros.

In addition to this northerly extension of granite, there are several smaller areas in the central and southern parts of the district. About $\frac{1}{3}$ mile southwest of Leopold there are outcrops which extend 1,200 ft. in an east-west direction and 600 ft. north and south. Just south of Leopold is another area of about the same size. To the west of Leopold at a distance of a little less than $\frac{1}{2}$ mile is a third one of these "islands," but this occurrence is very much smaller than the first two mentioned, its greatest dimension being not more than 250 ft. Besides these, a single small outcrop appears 300 ft. northwest of the Sampson shaft, on the hill back of the old Burro Mountain Co. office.

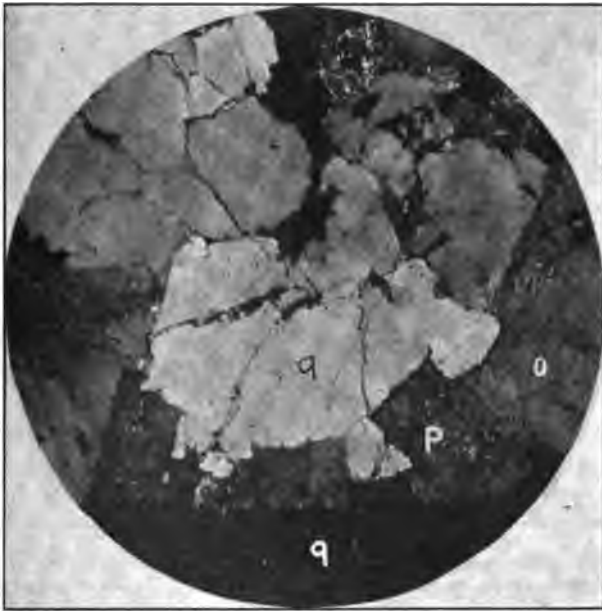


FIG. 7.—QUARTZ (q), ORTHOCLASE (o), AND PLAGIOCLASE (p) IN GRANITE.
CROSSED NICOLS. $\times 20$.

While slight textural and mineralogical variations are to be noted in the granite, there is no regularity in the distribution of the different phases. The age of the granite is pre-Cambrian, since it is found overlain by Cambrian sediments in some parts of the Silver City quadrangle.

3. Quartz Porphyry

Megascopic Characters.—The quartz porphyry is a light-colored porphyritic rock which always contains hexagonal phenocrysts of quartz,

varying in size up to 1 cm. in diameter of cross-section, and usually very striking phenocrysts of pink feldspar which are sometimes as much as 4 cm. long. The quartz phenocrysts show clearly their double-ended hexagonal pyramids, with edges somewhat rounded, and the pink feldspars have very well defined crystal outlines. In addition to these, smaller phenocrysts of a whitish feldspar are very common, and a few greenish grains of chloritized biotite were noted. In some places the white feldspars make up the background for the phenocrysts, but in others there is an aphanitic, blue-gray to white groundmass.

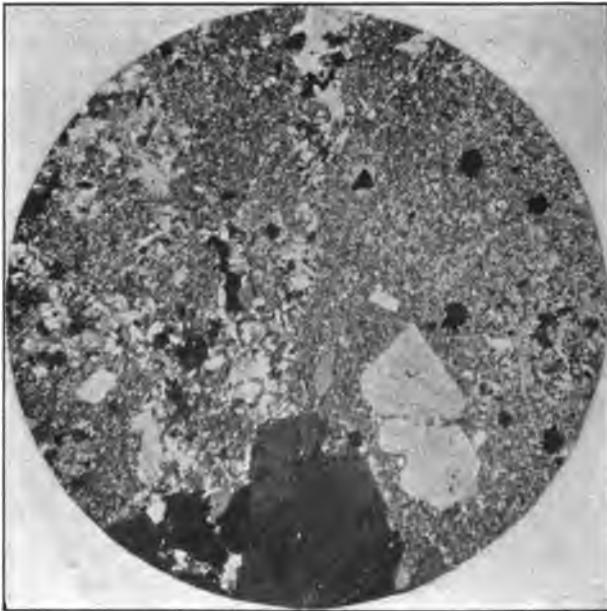


FIG. 8.—QUARTZ PHENOCRYSTS IN SILICIFIED QUARTZ PORPHYRY. TYPICAL UN-ALTERED SECTIONS SHOW MUCH MORE FELDSPAR. CROSSED NICOLS. $\times 20$.

Microscopic Characters.—Examined in thin section, the pink feldspar proves to be orthoclase, and the white, oligoclase. The notably rounded outlines of the quartz phenocrysts are clearly shown (Fig. 8). The groundmass is an aggregate of oligoclase, orthoclase, and an occasional grain of quartz. Apatite, zircon, and magnetite are the accessories present.

Structure, Distribution, and Age.—The quartz porphyry is intrusive into the granite. A large dike of it outcrops about 1 mile west of Leopold, just beyond Copper Mountain, and another one is found on the hill about $\frac{3}{4}$ mile northwest of Leopold, above the Leopold-Tyrone road. A few smaller dikes are found both underground and on the surface

in the vicinity of Tyrone, and in connection with the "islands" of granite to the south and east of Leopold.

The quartz porphyry is considered to be an earlier intrusion than the monzonite, since, while it is found in the granite, no occurrences have as yet been discovered where it cuts the monzonite. Furthermore, it is often mineralized. Its exact age is indeterminate, but the similarities in composition, structure, and texture between the quartz porphyry and the monzonite suggest the likelihood that the former did not precede the latter by a very long time, geologically, and that it was probably an earlier intrusion from the same magma.

4. Quartz Monzonite

Megascopic Characters.—The monzonites are extremely variable in texture. The most common phase about Tyrone and Leopold is a light-colored porphyritic rock of medium grain, containing abundant phenocrysts of white feldspar, and rarely a grain of quartz or biotite, in a dense whitish groundmass.

Some of the monzonite outcrops are of fine-grained porphyry rather than medium grained, and consist of a dense gray groundmass through which are sprinkled a few indistinct phenocrysts of feldspar.

Occasionally, however, there is found a phase which is almost granitic in texture. This contains an abundant whitish feldspar that shows striations and averages about 0.5 cm. in diameter; rather common grains of biotite which often show chloritization distinctly even in the hand specimen, some small grains of pinkish feldspar, and a little quartz which has the dull luster of an almost cryptocrystalline aggregate rather than the clear glassy luster of a single crystal. Whenever the texture of this rock tends toward porphyritic, the pink feldspar and the quartz make up the groundmass.

Different from any of these varieties is a rock which is found in narrow dikes in the granite in the northern part of the area. This rock is fine grained and dense, and wholly lacking in phenocrysts of any size. In the altered condition in which it is found, it is so quartzose and brittle in some occurrences that its resemblance to quartzite is marked. In other outcrops the quartzitic appearance may be less prominent, and at the opposite extreme are very dense phases which are so soft and smooth that they seem to be almost wholly lacking in quartz.

Microscopic Characters.—Under the microscope, the porphyritic varieties of the monzonite show large amounts of orthoclase and plagioclase, the latter, if anything, being slightly in excess. It does not predominate sufficiently, however, to change the identity of the rock. The plagioclase varies from oligoclase to andesine, the more basic phases being found to the south, away from the contact with the granite, which may

possibly be due to assimilation by the monzonite of blocks of the slightly more acid granite near the contact of the intrusive.

Quartz is present in practically all the monzonite, classifying it, therefore, as a quartz monzonite. It is very seldom present, however, in phenocrysts, but is rather to be found in small grains in the groundmass. Even when the texture of the monzonite is so coarse that it appears granitic in the hand specimen, the quartz is shown by the microscope to maintain its fine texture, filling in the interstices between the idiomorphic crystals of feldspar in the form of a fine aggregate, rather than in single grains that are nearly the same size as the feldspars. This feature is shown in Fig. 9 and is responsible for the rather dull luster of the

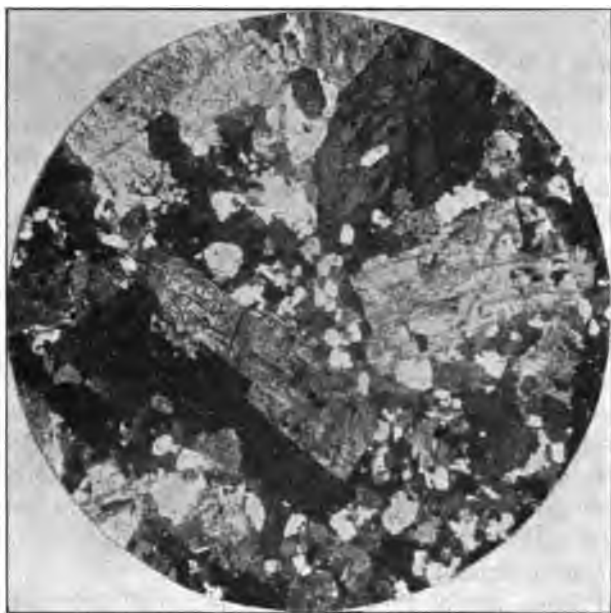


FIG. 9.—QUARTZ (SMALL GRAINS) AND FELDSPAR IN MONZONITE. CROSSED NICOLS. $\times 20$.

quartz as it shows in the hand specimen in these coarse varieties. It also furnishes a very good means of distinguishing the coarse monzonites from granite (Fig. 7).

The porphyritic monzonites contain apatite, zircon, and perhaps magnetite as common accessories. One section shows a small cluster of rutile needles.

The fine-grained dike rocks are found in thin section to be made up of quartz and secondary sericite (Fig. 10), no unaltered specimens having been obtainable. The phases which resemble quartzite in the hand specimen, show clearly in thin section an abundance of quartz in irregular

grains, together with many flakes of sericite. The quartz grains are intergrown with each other wherever they lie in contact. When they are not in contact, the spaces between them are filled with flakes of sericite, which form definite patches whenever the sericite is plentiful enough, and represent feldspars that have been altered. Quartz is relatively more abundant than the space occupied by the original feldspar, and even though some of it may be secondary, the rock as a whole was much more acid in character than the porphyritic monzonites.

The smoother phases are also made up of quartz and secondary sericite, but the patches of sericite are predominant and in extreme cases there is probably little or no primary quartz present. On the whole, their composition must have been more like that of the porphyritic monzonites.

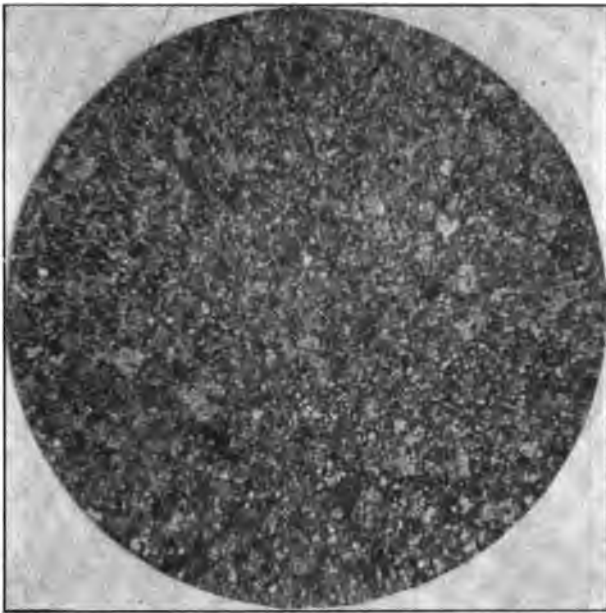


FIG. 10.—FINE-GRAINED MONZONITE. GRAINS OF QUARTZ AND FLAKES OF SERICITE. CROSSED NICOLS. $\times 50$.

The texture of these dike rocks is exceedingly fine. In the quartzitic phases, about 0.13 mm. is the maximum diameter noted for a quartz grain, and the average piece of quartz is about 0.05 mm. in diameter. One section, with quartz very predominant, showed a flow structure, and an average size of grain of about 0.006 mm.

Structure, Distribution, and Age.—The quartz monzonite is the main intrusive in the granite. The exposure of it in the area mapped in Fig. 6 is, however, but a corner of the larger mass, which stretches away to the

southwest for about 6 miles, with an average width of about 3 miles. The intrusive nature of the monzonite mass is clearly shown by the dike-like character of some of its offshoots, the increasing fineness of texture in the monzonite as the contact is approached, and the slight baking which has been effected in the granite directly at the contact.



FIG. 11.—FINE-GRAINED MONZONITE DIKE (a) IN BADLY ALTERED QUARTZ PORPHYRY (b). NATURAL SIZE.

A part of the contact between the granite and this larger monzonite intrusion crosses the mineralized area (Fig. 6). It is characterized by tongues of the monzonite extending irregularly into the granite. Although the islands of granite about Leopold are mapped with rather

even contacts, it is probable that much more irregularity would be shown if the surface débris were sufficiently stripped to show bedrock. This is strongly suggested by such exposures as there are.

With the exception of the very fine-grained phase, the differently textured varieties of the monzonite show little regularity of distribution. The medium-grained porphyritic monzonite is the most abundant. A very coarse monzonite has been found in the Old Virginia shaft, about $\frac{1}{2}$ mile east of Leopold. At a similar distance south of Leopold the coarse unmineralized phase of the main mass first begins to appear. As any contact with the granite is approached, the finer variety of the porphyritic monzonite is usually found.

The dikes of aphanitic texture, which are found cutting the granite in the northern part of the area, require separate consideration. While these also intrude the quartz porphyry (Fig. 11), they seldom, if ever, cut the porphyritic or granitic monzonites. The writer believes that they represent that part of the monzonite magma which intruded the granite country rock in small dikes, either at the time of the main intrusion, or perhaps soon after, probably accompanied more or less by water vapor, and hence much after the fashion of the aplitic intrusions of a granite batholith.³ The name "aplite," however, is hardly applicable to all of these rocks. Many of them must, when fresh, have a composition very similar to that of the coarser monzonites, and they should therefore be called fine-grained monzonites. A few of the more quartzitic varieties may have the composition of true aplites, and they may be properly given the name aplite.

The age of the quartz monzonite is post-Cretaceous, as determined by Paige in his study of the Silver City quadrangle.⁴

5. Volcanic Breccia

Description.—This rock is a red porous clastic, consisting of fragments of varying sizes with 10 cm. as the maximum diameter. The fragments are quite angular and are composed mostly of red porous lava with occasional pieces of granite or monzonite. The breccia owes its red color to the fact that it is very thoroughly soaked with hematite and limonite which have been diffused by weathering of the lava fragments.

Structure, Distribution, and Age.—A few small outcrops of breccia are found about 1 mile south-southeast of Tyrone, near the road to the Savannah property. They are only a few feet thick and are the remnants of a formation which was once much more extensive than it is now. Other outcrops are to be found in the Little Burros. The outcrops observed to the south of the Mangas are stratified and have a north-east

³ Daly, R. A.: *Igneous Rocks and Their Origin*, p. 368 (1914).

⁴ *Bulletin* No. 470, U. S. Geological Survey, p. 133 (1911).

strike with a dip of 15° to the northwest. The pronounced red color in the vicinity is due to the large amount of iron oxides which the breccia contains.

The age of the breccia is presumably Tertiary.

6. Gravels

Description.—The gravels are made up of varying-sized fragments of all the older rocks of the district. The grains and pebbles are slightly rounded.

Structure, Distribution, and Age.—The gravels represent the débris worn from the other rocks of the area in Pleistocene and Recent times and washed into the valleys. Their distribution and rather angular form indicate their transportation and deposition by the few, but torrential, rains of an arid region, and this process is still going on.

They are sufficiently definite in form and composition to be a geologic formation, and they occupy the lower levels in the region, chiefly the Mangas Valley. Their southerly boundary is quite distinct and is shown on the map.

III. ECONOMIC GEOLOGY

1. Introduction

The copper of the Burro Mountains occurs in the mineral chalcocite, which has been produced by downward secondary enrichment. Chalcocite replaces primary pyrite, for the most part, and hence it is found, like the latter, as vein material, along with quartz, and as disseminated replacements in those portions of the rock bordering the fissures. The fractures, both large and small, which these minerals have filled, occur in a well-defined zone. Vertically, the three zones, of oxidation, downward secondary enrichment, and primary sulphides, are well marked, but their thicknesses and depths below the surface are very irregular.

2. Mineralogy

The primary vein and replacement minerals of the district are quartz, pyrite, chalcopyrite, and sphalerite. The secondary minerals resulting from (a) the processes of superficial alteration and downward enrichment are chrysocolla, malachite, azurite, cuprite, native copper, limonite, hematite, and chalcocite; (b) from hydrothermal action upon the rocks, sericite, chlorite, and epidote; and (c) from the action of descending meteoric waters on the rocks, kaolinite.

Quartz.—This mineral is not only present in the district as an original constituent of the igneous rocks, but it also occurs as a secondary mineral in three different forms, viz.:

(1) As a gangue mineral of the ores, where it is found in crystalline grains of moderate or small size in fractures, or saturating the country rock (Figs. 23 and 24). In this form it is either glassy or milky white, is found throughout the area, and has been deposited from ascending solutions.

(2) As a precipitate, separated out during the sericitization of the feldspars. This variety is not very abundant and is much less distinct than gangue quartz, but may occasionally be noted in thin section.

(3) As a weathering product, descending meteoric waters having reworked the earlier quartz more or less, and in some cases left it deposited in the upper portions of veins, as a crypto-crystalline aggregate, or chalcedony. This is often stained deep red with hematite, and in such condition may resemble jasper. It is found in the northern part of the district. The presence of hematite instead of limonite is interesting and corresponds with the observations of Emmons⁵ and others that hematite is apt to be the ultimate result of oxidation in place of a hydrous oxide, where the climate is arid.

Pyrite.—Pyrite is by far the most abundant primary sulphide of the district, and together with quartz makes up practically all of the original vein and replacement material. It is usually granular, and crystal faces or outlines are not very commonly seen. It is found throughout the mineralized area.

Chalcopyrite.—This mineral is distributed widely in the district but in very small amounts, and in such minute specks that it cannot, as a rule, be seen in the hand specimen, but its presence can be detected by microscopic examination of polished sections (Fig. 22). It may occur in pyrite or in quartz gangue, and is undoubtedly an original mineral. It appears to be very unevenly distributed.

Sphalerite.—Sphalerite was found in very few of the many specimens examined, and seems to be extremely rare, but, like chalcopyrite, it is present in such minute grains that its existence can seldom be noted except with the microscope (Fig. 17). There seems to be no doubt that it is of primary character, occurring with the pyrite and chalcopyrite.

Chrysocolla.—This hydrous copper silicate is the most abundant oxidized copper mineral found in the camp, and is chiefly responsible for the blue color of the outcrops at Copper Mountain, as well as at the Boston, No. 1, St. Louis, and McKinley shafts, etc. It not only occurs scattered irregularly through the surface portions of the rocks, but also may be found in small, clean-cut veins that are made up wholly of this mineral (Fig. 13). These are too free from quartz to represent former veins that have been changed to chrysocolla, but are probably later fractures in which solutions have deposited the hydrous copper silicate.

⁵ Emmons, S. F.: Los Pilaes Mine, Nacozari, Mexico, *Economic Geology*, vol. i, No. 7, p. 633 (July-Aug., 1906).

The chrysocolla is colored various shades of blue and bluish green when pure.

A black, pitchy material is occasionally associated with the blue-green varieties of chrysocolla, as, for example, in the large fissure about $\frac{1}{3}$ mile southwest of Leopold. This contains manganese, and is probably copper pitch, or the German *Kupferpecherz*, a variety of chrysocolla stained brown or black with manganese oxide, which is well known at Globe.⁶ It is deeper black in color than much of the material from Globe.

Malachite.—Malachite is occasionally found in the district, though not in any abundance. At Copper Mountain it occurs in veinlets and coatings more or less intermixed with chrysocolla. It is fibrous, but the fibers are seldom more than 2 mm. in length.

Azurite.—Deep blue azurite occurs in very small quantities at Copper Mountain, but is rarely seen in any other part of the district. On the whole it is much more uncommon even than malachite. It is in small, irregular vein-like masses, and usually shows a thin streak of malachite running through the center of it.

Cuprite.—This mineral has been reported from the district,⁷ but did not come under the observation of the present writer.

Native Copper.—A few very small flakes of native copper have been found in monzonite in the Niagara tunnel about 3,000 ft. from the portal. They occur in very small and irregular fractures, and are usually accompanied by a red powdery hematite.

Limonite.—Limonite is very abundant as a surface stain and a filling of fracture spaces in the zone of oxidation. It is found throughout the mineralized area with the exception of a few localities where the oxidized form of the iron is hematite, and its quantity varies directly with the intensity of the primary pyritic mineralization. It shows a pronounced tendency to leach out on to the surface of the outcrops and talus fragments, thereby covering the surface with a thin brown film and giving the district its consistently rusty appearance.

Hematite.—In the northern part of the area, hematite seems to take the place of limonite to some extent in surface outcrops. It is red and finely divided and occurs either as a powdery material, or disseminated as a coloring matter in quartz. It is also found in considerable quantity in the volcanic breccia in the northeastern part of the district.

Chalcocite.—As at Butte, both massive and sooty varieties have been found in the Burro Mountains.⁸ No crystals, however, have been

⁶ Ransome, F. L.: Geology of the Globe Copper District, Ariz., *Professional Paper No. 12*, U. S. Geological Survey, p. 123 (1903).

⁷ Lindgren, Graton and Gordon: Ore Deposits of New Mexico, *Professional Paper No. 68*, U. S. Geological Survey, p. 322 (1910).

⁸ Sales, Reno H.: Ore Deposits at Butte, Mont., *Trans.*, xlv, 49 (1913).

noted. The massive chalcocite is dark steel gray in color, and very homogeneous. It is by far the more common. The porous, sooty variety is occasionally seen where the massive chalcocite has been affected by meteoric waters in the upper parts of the chalcocite zone. Chalcocite shows well in polished sections (see Figs. 17 to 19). Its distribution is one of the main considerations of the district and will therefore be taken up under a later heading.

Sericite, Chlorite, and Epidote.—As will be discussed below, sericite, chlorite, and epidote have been developed by hydrothermal action upon the feldspars and ferro-magnesian minerals in the country rocks. The first is very abundant, the second rather uncommon, and the third rare.

Kaolinite.—Kaolinite has been formed from the non-metallic minerals of the rocks by meteoric waters, as will be noted in a later portion of this paper.

3. Occurrence of the Ores

The ores occur in two forms: (1) as the filling of open spaces, such as fissures of varying size and the interspaces of breccia; and (2) as disseminated grains in the walls adjacent to these open spaces.

(1) *Filling of Open Spaces.*—The district is characterized by a rather well-marked zone of fracturing, the surface outcrop of which is roughly triangular in form, with its apex toward the southwest, its sides on the west and southeast (see map, Fig. 6). Within this triangular area there are fractures of all sizes, from fissures that are prominent both on the surface and underground, to minute breaks that are visible only under the microscope. The fractured zone lies partly in granite and partly in monzonite, and shows some rather definite characteristics.

Larger Fractures.—The larger fractures, or "slips," as they are called in the Burro Mountains, which vary from a fraction of an inch to a foot or more in width, are usually rather straight, and approximately parallel to each other over small areas. Their spacing varies from several inches to a few feet, and the dips are mostly steep.

The larger slips are often quite continuous and sometimes a single fissure may be followed for a considerable distance either on the surface or in the underground workings. More often, however, there appears to be a zone made up of two or more of these strong fractures accompanied by the usual large number of smaller breaks, and this zone may persist for 1 or 2 miles.

The slips at shaft No. 1 are good examples of large persistent fractures and all along the western side of the fractured area from this shaft to Copper Mountain the fissures form a well-marked, though somewhat irregular, zone. About $\frac{1}{2}$ mile southwest of Leopold there is a strong slip which is part of a zone that can be traced for nearly a mile across the country in a direction about 10° north of east. Likewise the southern-

most fissure plotted on the map is a part of a distinct zone extending to the northeast, and readily traceable on the surface from one end to the other.

It is an important fact, however, that only along the sides of the area do we find fissures or fracture zones of one general strike, with comparatively few and unimportant cross fractures. In the center of the area, as, for example, around Leopold and Tyrone, there are several fracture zones, which cross each other at various angles. These fissures are fully as strong as those along the boundaries, but not quite as persistent. Furthermore, some of them are not as straight, such as the so-called "St. Louis slip," for instance, which is shown at the St. Louis shaft. Southward from the shaft it has a general northeasterly strike, but on the north it curves until it has an easterly strike, which it holds until it has crossed the gulch in which Leopold is situated, when it turns back to a northeasterly trend. The most important direction of cross fracturing is roughly north and south, as for example in the immediate vicinity of Leopold; and again, near the Niagara and Boston shafts. The broad result of the fracturing, then, seems to have been a generally broken condition along the center of the area grading into a few consistent fissures along the sides. The latter do not cease abruptly as the distance from the center increases, but gradually diminish in size and number until solid rock is reached.

Two important structural characteristics of the fractured zone are defined by the larger fractures. In the first place, there is an apparent convergence of many of the slips toward an imaginary point about 2 miles southwest of Leopold. This may be noted on the map and is most pronounced toward the lateral boundaries of the area, where cross fracturing is uncommon. It is less marked in the center, where fissures of several systems are present, but even there it appears to be a strong tendency. On the north and northeast the gravels cover the continuation of the fissures, and thereby conceal their structure in that direction. At the eastern corner of the fractured zone, however, the fractures seem to be branching and fraying out, and it is quite likely that this structure is continued to the north.

Another noteworthy tendency is in the directions of dip taken by many of the slips. If the angle which the fractures make by their convergence be bisected, most of the fissures in the western half that are not vertical dip toward the east and southeast, while those in the southeast half show a tendency to dip to the northwest. This, of course, refers to the majority, but nevertheless the exceptions are comparatively few.

It is seldom possible to say whether or not there has been movement along the fractures. Since the rocks are massive, the displacement of strata cannot be used as a criterion, and recourse must be had to the less often observed evidences of displacement of dikes and cross fractures,

presence of gouge, and slickensides. Most of the fractures show none of these, and there is no indication of a big fault such as might cut off an entire rock formation, or leave a marked escarpment on the surface. On the other hand, some faulting is undoubtedly present. Slickensides are occasionally seen, a good example being in an outcrop on the road just south of Tyrone. Gouge may be found in a few slips, the St. Louis for instance. Small displacements of fissures may be observed, as in one of the Leopold orebodies, where such movements were noted as a 1-in.



FIG. 12.—LIMONITE-FILLED VEINLET IN FINE-GRAINED MONZONITE, SHOWING SMALL DISPLACEMENT. NATURAL SIZE.

fissure displaced 6 in. Very small displacements are shown in hand specimens from shaft No. 1 (Fig. 12). The evidence, therefore, points to the existence of small and irregular faults, perhaps in abundance, with big movements very doubtful and as yet unproved in any given instance.

Another question is in regard to the relative ages of fissures of different strikes. That some of the fractures are younger than others is clear in many instances. The small displacements noted above are cases in point. In the writer's opinion, however, generalizations regarding

the different directional systems of fractures are impossible. No one system is distinctly older or younger than another.

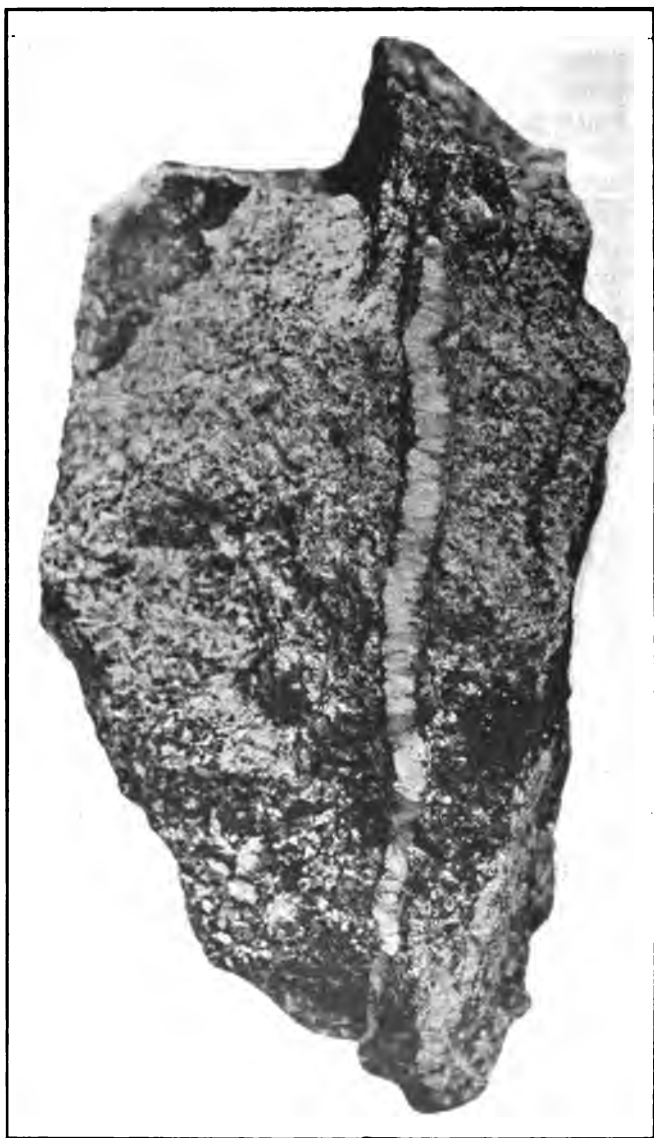


FIG. 13.—VEIN OF PURE CHRYSOCOLLA IN MONZONITE. NATURAL SIZE.

Smaller Fractures and Breccia.—Besides these larger breaks with their more or less consistent strikes and dips, there are very many small fractures, never more than a few inches in length, and often the merest frac-

tion of an inch in width. They have no regularity in direction or length. Their abundance depends on the brittleness of the rocks, and while in most of the monzonites they appear simply as frequent hair-like cracks, in the brittle granite they are much more abundant, and sometimes have developed to such an extent that breccias are the result. The breccia fragments vary in size up to about 1 ft. for the largest dimension, and may be found in small masses or in large bodies of lens-like form several hundred feet in length. That the brecciation is due to the brittleness of the granite is well shown by the fact that dikes of monzonite passing through a mass of breccia, though affected by the same stresses, are not fractured to the extent of brecciation.

Brecciation has caused at least one very important orebody in the district, which has been opened up between the 300 and 400 ft. levels in the shaft No. 3 workings. In addition to this occurrence breccia is found at one or two other points underground, and also at several places on the surface.

Origin of the Fractures.—While at best only a speculation, nevertheless it may be worth while to formulate a theory for the origin of these fractures.

They could not have been caused by contraction of the monzonite on cooling, since they are found only along a small portion of the monzonite mass.

They may possibly belong to a shear zone, but this seems rather improbable, for two reasons. In the first place, a shear zone of such great width and length would be expected to show much greater evidences of displacement than are found in the Leopold-Tyrone area. In the second place, it is very difficult to imagine a closed shear zone in which the fractures in each half dip toward the center. A fairly consistent dip in one direction would be expected.

The writer believes that this fracture zone can be explained by assuming a caving of the surface due to the extrusion of molten material from immediately beneath. There has been igneous activity all about this district in fairly recent times, and especially later than the monzonite there has been the intrusion of stocks of rhyolite which are found directly across the Mangas Valley from the fractured area.⁹ The fracturing was caused by the movement of this rhyolite, and probably commenced some time before the rhyolite became solidified in its present position. It continued, however, after the solidification of the rhyolite, since a few fractures are found in the latter, and their northeast strikes indicate that they are continuations of the fractures to the south of the Mangas.

With this hypothesis as a basis, the characteristic features of the fractured area can be readily accounted for.

⁹ Paige, Sidney: *Bulletin No. 470, U. S. Geological Survey*, p. 133 and map (1911).

A diagrammatic cross-section of such an area is shown in Fig. 14. In the first place, the fractures along both sides of the area would not be vertical, but would dip toward each other, since caving involves an area of surface larger than that of the cavity into which the rocks have fallen. In mining operations it has been found that in hard rocks the outermost fractures dip at angles of from 60° to 75° , and these correspond with the angles of dip noted in the vicinity of Leopold and Tyrone. The central fractures would show more nearly vertical dips.

While the majority of the larger fissures might dip as described above, there would, nevertheless, be many cross fractures and fracture zones, which would break the rock into blocks, especially in the center of the caved area. Along the edges the tendency might be to develop fissures in a single direction, but toward the center a generally broken condition

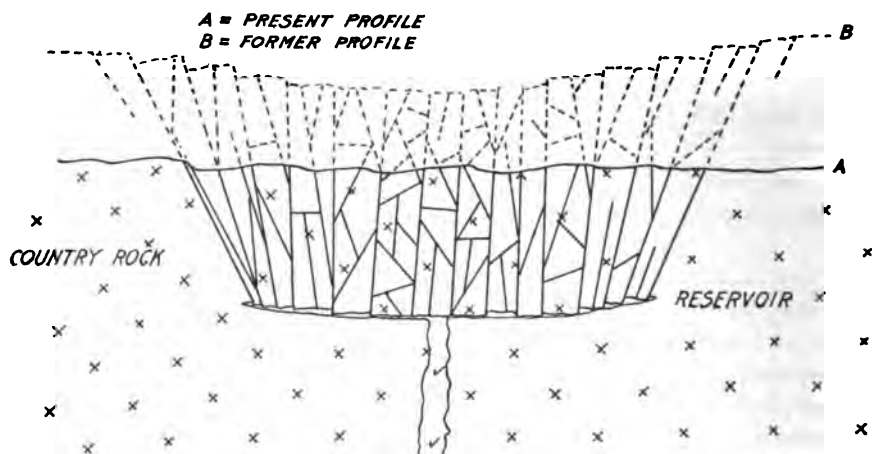


FIG. 14.—DIAGRAMMATIC CROSS-SECTION OF FRACTURED ZONE.

of the rocks, with more fractures, and more directions of fracture, would be produced.

Furthermore, innumerable smaller fractures would be developed, increasing in number with greater brittleness of the rock, until in extreme cases a breccia might result. This is the explanation of the breccia found at Tyrone, and the brittle rock is the granite, while in the tougher monzonite breccias are seldom, if ever, found.

Finally, the actual caving would consist of a settling of the blocks formed by these breaks and a readjustment to the new conditions, which would cause a small amount of displacement along nearly all the fissures, without relatively great movement in any case. And while all the fractures would not be of exactly the same age, on the other hand, no consistent differences in age between different systems of fractures would be expected.

Mineralization of the Fractures.—In these fractures, solutions have deposited quartz, pyrite, chalcopyrite, and sphalerite, and secondary processes have enriched portions of them to an extent sufficient to make their mining profitable. But in addition to the ore which is found in this multitude of fractures, there is an important amount found disseminated in their walls.

(2) *Disseminations.*—Primary pyrite has been deposited in the walls of the visible fractures in grains which range in size from exceedingly small particles up to those which are perhaps 5 mm. in diameter, with about 1 mm. as the average.

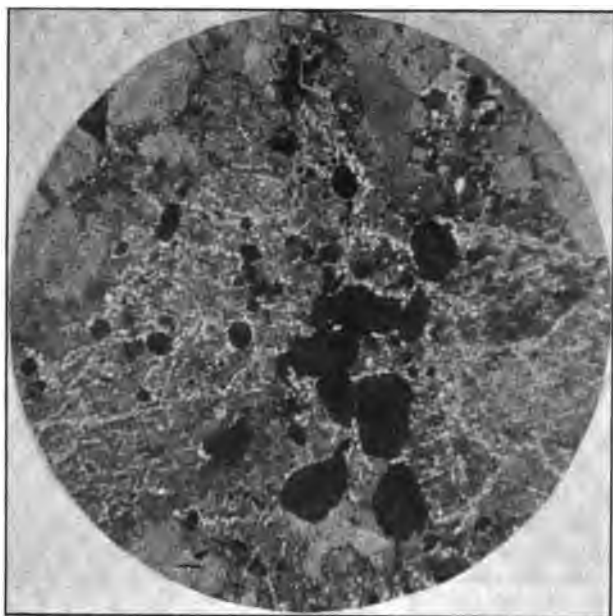


FIG. 15.—PYRITE (BLACK) REPLACING SERICITIZED FELDSPAR. LIGHT STREAKS ARE SERICITE. QUARTZ GRAINS AROUND THE EDGES. CROSSED NICOLS. $\times 50$.

Upon examination of thin sections of the ores, it is found that these grains were deposited in two ways: (1) as filling of minute open spaces; and (2) as metasomatic replacements.

The former occupy the irregular fracture spaces of microscopic size that exist in the rocks, and are of course similar in mode of origin to the more evident open-space fillings, the distinction between them being simply an arbitrary difference in size.

These micro-fractures develop commonly between the mineral grains, and also throughout the feldspars, as in the case of the granites, but to a much less degree in the quartz. The same is true in the mon-

zonites, but with the quartz more subordinate the fractures are more evenly distributed through the rock.

Secondary quartz is often present with the pyrite, and thin sections show all stages of gradation, from the smallest fractures with only one or two grains in them to the more distinct veinlets, where the filling is a granular aggregate of quartz and pyrite.

The second class of disseminations, however, have been formed strictly by metasomatic substitution, and they appear under the microscope to be for the most part replacements of the feldspars (Fig. 15). Clearly

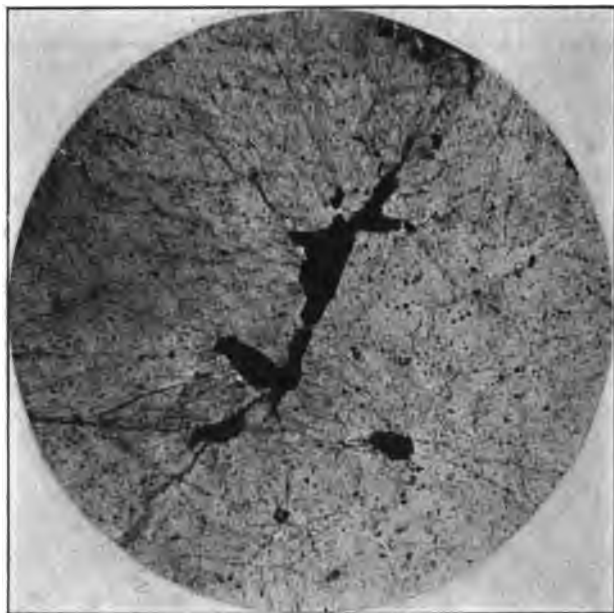


FIG. 16.—PYRITE (BLACK) REPLACING QUARTZ OF THE GRANITE. SHOWS FRACTURES THAT SERVED AS CHANNELS OF ACCESS. CROSSED NICOLS. $\times 50$.

defined cubes of pyrite are often developed, and some grains show a connection with small fractures which have served as channels of access.

Many grains show the characteristics of both types, in that they are fracture fillings extended by replacement. Each of these occasionally appears to replace the quartz as well as the feldspars (Fig. 16).

The disseminations are scattered through the walls of the fissures for varying distances, depending upon the openness of the fracturing and therefore the freedom with which circulating waters could penetrate the rock. The walls are usually more or less impregnated from one slip to the next, and therefore the actual distance can seldom be measured.

4. Oxidation and Downward Enrichment

The value of these deposits as a source of copper is due to the presence of a definite, though irregular, zone of secondary chalcocite of sufficient extent and richness to work.

Oxidized Zone.—Between the secondary chalcocite and the surface there is a zone of oxidation which is irregular in depth, and from which the copper has been very thoroughly leached over most of the area. The pyrite which formerly existed in the fractures and disseminations has been changed to limonite, and the rock itself has a bleached appearance.

The depth to which leaching has extended is extremely variable. The fissures are leached much deeper than the less pervious rock between them, indicating that the freedom with which surface waters may reach considerable depths is a big factor in determining the vertical distribution of the different zones. It is not at all uncommon to find the slips thoroughly leached while the disseminations in their walls are enriched.

Furthermore, steep fissures are apt to be leached deeper than the flat ones. It is occasionally found that where a flat slip crosses a steep one, the latter is leached, whereas the former is enriched, which the writer would attribute, not to any difference in age between the two, but rather to the fact that a steep fissure would allow freer and deeper circulation of surface waters than would a flat one, with consequent deeper leaching.

In a few places some copper remains in the oxidized zone. It is in the form of chrysocolla and malachite, with some azurite and rare occurrences of cuprite and native copper in very small amounts. Such is the case at shaft No. 1, at the Boston, St. Louis, and McKinley shafts, and at Copper Mountain, as well as in a few other scattered areas. With the possible exception of Copper Mountain, there is hardly enough oxidized copper at any place to be workable.

The occasional presence of copper minerals in the oxidized zone is probably due to the impermeability of the rocks in these localities, which has hindered a free downward circulation of meteoric waters, and thus forced the precipitation of copper within the zone of oxidation, chrysocolla having been the result to a much greater extent than malachite or azurite because of the scarcity of carbonic acid, and the presence of silica under conditions suited to the formation of the silicate.¹⁰ The cause of this impermeability is not always clear. It may be due to the tightness of the original fracturing, or to an unusual amount of silicification. In a few cases it would appear that dikes of the more impermeable quartz porphyry have been responsible.

¹⁰ Kemp, J. F.: Secondary Enrichment in Ore-Deposits of Copper, *Economic Geology*, vol. i, No. 1, pp. 24 to 25 (Oct.-Nov., 1905).

Enriched Zone.—Below the oxidized zone is a zone of secondary enrichment in which the mineral developed is chalcocite.

The chalcocite is metallic or dull in luster in most cases, though occasionally sooty where recent changes in water level have allowed oxidizing waters to come in contact with it and by partial solution convert the massive chalcocite into the porous, sooty form.¹¹ Although the ores have been thoroughly examined under the microscope by means of reflected light, no secondary copper sulphides, other than chalcocite, have been disclosed.

The chalcocite has been developed by replacement of primary pyrite, chalcopyrite, and sphalerite. Sphalerite is rare, but nevertheless, as shown in Fig. 17, it is easily replaced by chalcocite. Although this has not been as generally observed in the studies of other ore deposits as has the replacement of pyrite and chalcopyrite, it is not at all unknown, and is clearly shown in the Burro Mountain specimens.

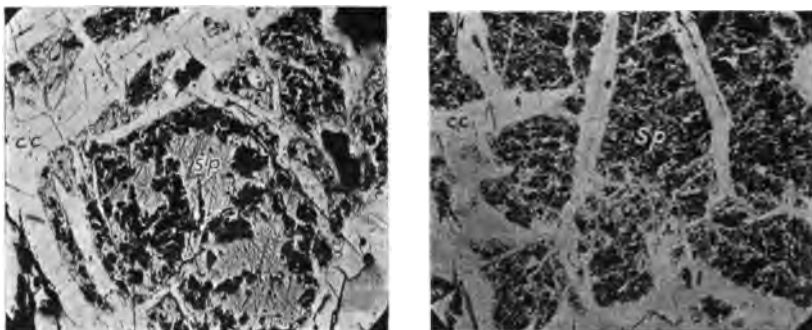


FIG. 17.—SPHALERITE (sp) BEING REPLACED BY CHALCOCITE (cc). BLACK SPOTS ARE PITS IN THE SURFACE. REFLECTED LIGHT. $\times 40$.

A complete gradation may be observed, from the first formation of a film about the original sulphide grain, with a thin veinlet or two running through it, to the point where there is nothing left but an occasional small residual core in a matrix of chalcocite, which indicates with shadowy outlines the size and shape of the replaced pyrite grains (see Figs. 18 and 19). No sulphides intermediate between pyrite and chalcocite have been noted.¹²

Since it therefore occupies the position of the primary sulphides, the chalcocite is found in fissures and fractures intergrown with quartz, and in disseminated grains in the adjacent walls (see Fig. 20). The ore as it is blocked out will average about 2 per cent. at Leopold and 3 per cent.

¹¹ Sales, Reno H.: *Trans.*, xlv, 49 (1913).

¹² Graton, L. C., and Murdoch, Joseph: *The Sulphide Ores of Copper*, *Trans.*, xlv, 40 (1913).

at Tyrone. The slips are much richer than this, but the poorer rock between brings down the value of the portion to be mined.

The chalcocite zone is irregular in thickness, with a maximum of 200 to 300 ft. It is found throughout the horizontal range of the mineralized area with the exception of one or two places where imperviousness of the rocks has caused a diffusion of the copper-bearing waters and consequent lack of a secondary zone.¹³ About 175 ft. southeast of shaft No. 2 is an instance.

The zone of enrichment does not show a close relation to the present water table. At the time of the writer's visit, this level was at a depth of from 275 to 400 ft., with the Leopold orebodies to a considerable extent above the ground-water level; while at Tyrone large masses of chalcocite are both above and below it.

Taken as a whole, however, the secondary zone shows a very regular

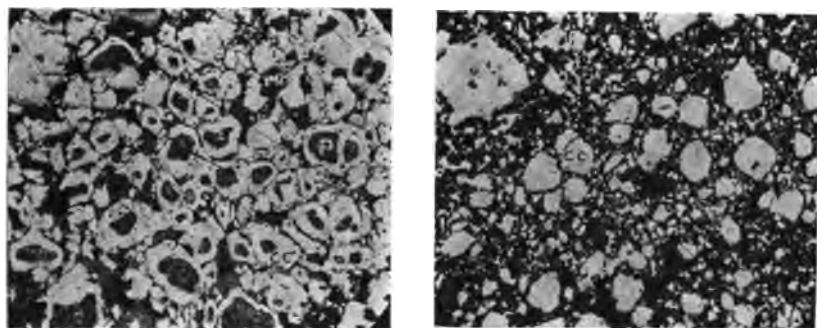


FIG. 18.—REPLACEMENT OF PYRITE (p) BY CHALCOCITE (cc). OUTLINES OF ORIGINAL PYRITE GRAINS INDICATED BY HOLLOWES AND GANGUE MINERALS, WHICH APPEAR BLACK. REFLECTED LIGHT. $\times 40$.

arrangement in the matter of depth from the present surface. It is quite near to the surface in the southern part of the area, as, for example, at Leopold, but becomes gradually deeper toward the northeast. When the Sampson shaft was sunk, chalcocite was found within a very few feet of the surface, and even along the larger fissures oxidation has not extended very deep. At Tyrone, however, the secondary zone is found at a considerable depth, and leaching has gone as deep as 700 ft. in places. Observations at intermediate points prove a gradually increasing depth from Leopold to Tyrone.

This tendency of the secondary zone, and its lack of correspondence with the present water table are due to the changes which have taken place in the relief, and consequently the water level, of the district.

In Fig. 21 the heavy line represents a profile from the Big Burros to

¹³ Penrose, R. A. F.: *Economic Geology*, vol. ix, No. 1, p. 20 (Jan., 1914); also Emmons, W. H.: *Bulletin No. 529, U. S. Geological Survey*, p. 20 (1913).

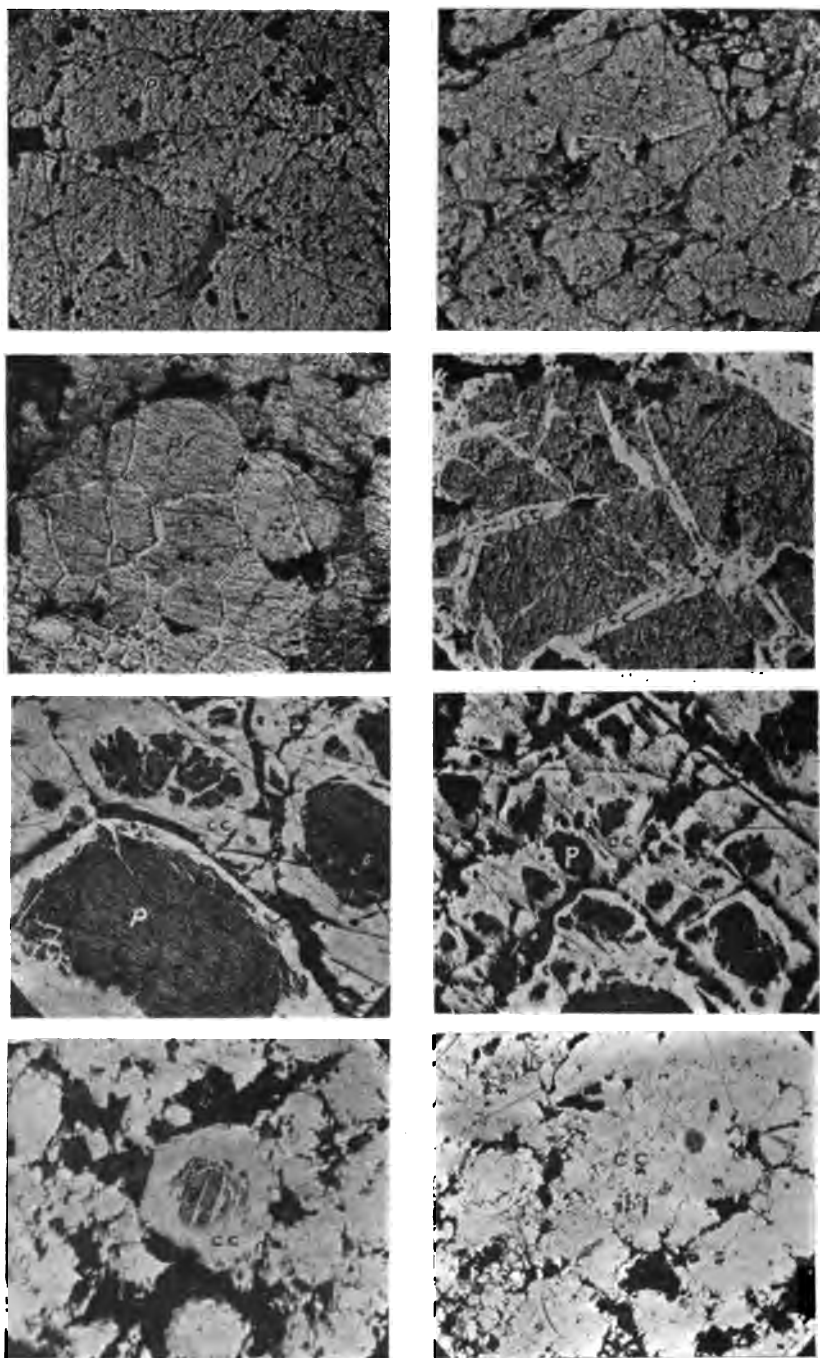


FIG. 19.—PHOTOMICROGRAPHS OF POLISHED SPECIMENS, ARRANGED TO SHOW PROGRESSIVE STAGES IN THE REPLACEMENT OF PYRITE (p) BY CHALCOCITE (cc). REFLECTED LIGHT. $\times 40$.

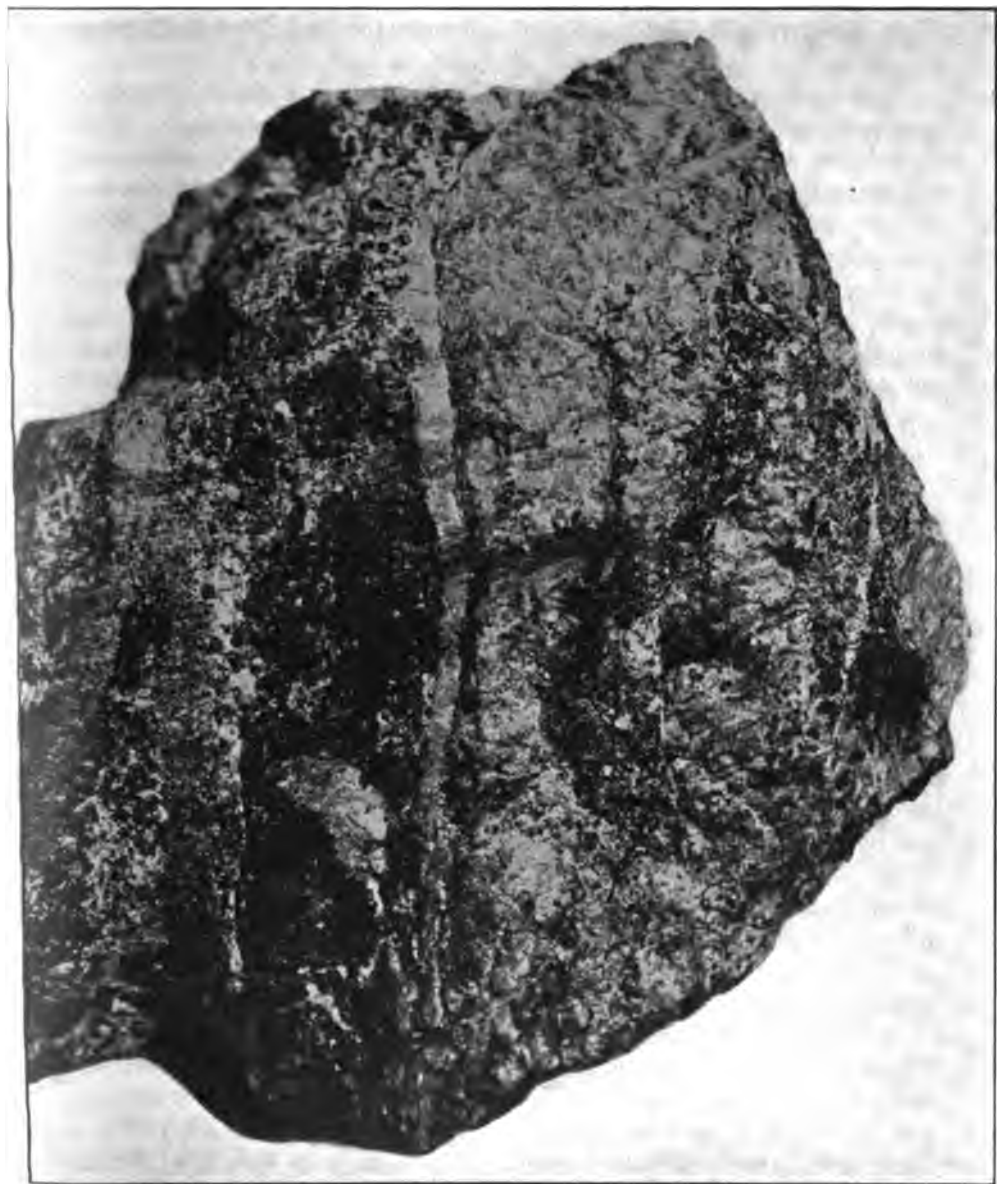


FIG. 20.—SPECIMEN OF ORE FROM SAMPSON SHAFT. SHOWS "SLIP" WITH LATER VEINLET OF QUARTZ. BLACK IS CHALCOCITE, WHITE IS QUARTZ. NATURAL SIZE.

the Mangas Valley, and on it are indicated the locations of Leopold and Tyrone. The broken line, *C*, represents the general profile of some former surface before erosion had worn the land down to its present level.

The location of this second profile is based on the tendency of erosion to wear down hills or mountains faster than valleys, especially in an arid region where so much of the work of erosion is done by infrequent but torrential rains that wash débris from the heights down into the larger valleys, but do not maintain constant flowing streams to erode these valleys very rapidly.

The Mangas Valley itself was probably initiated during a former period of erosion when the climate was less arid than it is now, or perhaps in the early part of the present period, under somewhat different conditions than those which prevail to-day. But for a large part

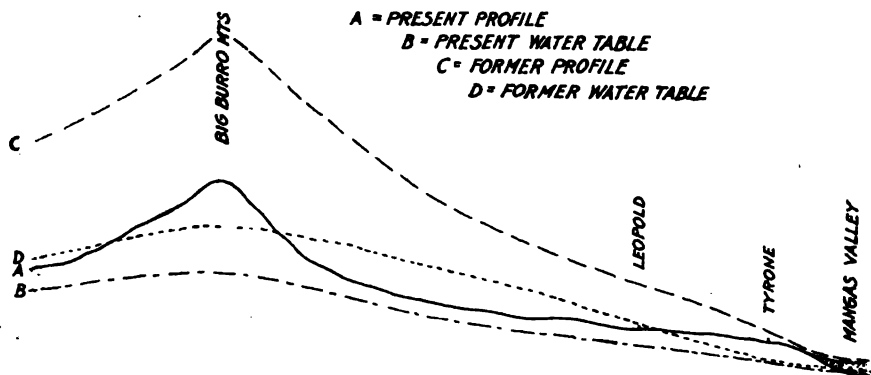


FIG. 21.—PROFILE OF BIG BURRO MOUNTAINS COPPER DISTRICT.

of the present stage of erosion the climate has been arid, as is proved by the angular form and undecomposed condition of the gravels, and the Big Burros have been worn down relatively fast, while the Mangas Valley has remained about where it was.

In this connection it is notable that the outcrops of many fissures about Tyrone contain, in place of the limonite and granular quartz which are found in the southern part of the area, a chalcedonic or crypto-crystalline variety of silica stained deep red with hematite. Chalcedony¹⁴ is a mineral that, under these conditions, would form very slowly, at or near the surface, and its presence in the more northerly veins indicates that their outcrops have been nearly stationary for a long time, whereas its absence to the southward indicates that in this section the veins have

¹⁴ Bulletin No. 529, U. S. Geological Survey, pp. 156 to 157 (1913).

been wearing away constantly without any part of them remaining long enough at the surface for chalcedony to develop.

Likewise, hematite stains this crypto-crystalline silica, but limonite is the surface form of the iron to the southward, which indicates that the northern outcrops have been longer exposed to the influence of the dry, hot climate than the more southern ones. These facts serve as additional evidence in favor of the second profile of Fig. 21.

Finally, the dotted line, *D*, represents the water table of such a former surface as that shown by the line *C*.

The important feature of the diagram is that the former water table crosses the present surface profile about at Leopold, but gradually diverges from it toward Tyrone. In a similar manner, any other former water level would pass comparatively near to the surface under Leopold, but at a considerably greater depth under Tyrone, and only recently has the water table been depressed below Leopold, while it has been as far below Tyrone as it is at present, during this whole stage of erosion. Hence, if the secondary zone is formed at or near the water level, it should be found at a depth of several hundred feet below Tyrone, because the water table has been at that depth for a long time. But at Leopold conditions have been different. Although the secondary zone tended to form at the water level, the latter has been sinking so fast that the former has been unable to keep up with it. This is because, in an arid climate, run off, with its accompanying erosion, is large, but absorption, and consequently alteration and enrichment, are small, so that erosion and the lowering of the water table go ahead of the downward migration of the secondary zone. In this way the chalcocite zone at Leopold has been left marooned above the water table and very near to the present surface. Going from Leopold to Tyrone, the water table has been considerably below the surface for an increasingly longer time, so that a better opportunity for leaching has been given and the depth of the chalcocite zone below the present surface gradually increases, while the amount of downward migration of this zone becomes less.

In the above discussion it is considered that the secondary sulphides could not have been precipitated consistently above the water level, because, on the whole, the fracturing of the Burro Mountains district is open and abundant, and there must have been a free enough access of oxygen and air to prevent any such precipitation.

Primary Zone.—The secondary sulphide at depth gives way to unaltered primary minerals. The line of separation between the two is neither regular nor sharp, and the depth at which it is reached depends on the permeability of the rocks. Locally, primary sulphides are found within a few feet of the surface, and unaltered cores are quite common throughout the deposit. On the other hand, the deepest workings, which

reach about 300 ft. at Leopold and 720 ft. at Tyrone, probably represent the lowest limit of any possible enriched zone.

Source of Copper.—The primary sulphides contain copper, although in very small quantities, and from the standpoint of a few years ago it would have been sufficient to say that the original pyrite was "cupriferous," without further qualification.

More recently, however, the work of Simpson at Butte¹⁵ and the broader studies of Graton and Murdoch¹⁶ indicate that, chemically, cupriferous pyrite does not exist, but that the copper in lean pyrite masses is present as a copper mineral scattered in very small amounts through the other sulphides.

The primary copper of the Burro Mountains furnishes an example of this. Chalcopyrite is found in polished sections of primary sulphides taken from the Bison orebody and from the workings of the Sampson, No. 1, No. 2, and No. 3 shafts. It is generally very scarce, is usually too small to detect even with a hand lens, and undoubtedly represents the source of the copper.

It occurs in small irregular grains, in veinlets, and in more or less rounded patches (Fig. 22). It is believed to be approximately contemporaneous with pyrite. When in grains, chalcopyrite and pyrite may occur as neighbors, with interlocking contacts, and neither mineral impressing its form upon the other. When in veinlets they may occur together in a similar manner, or they may be found in different parts of what appears to be one fracture. While the patches of chalcopyrite in pyrite at first seem to be quite rounded and drop-like, upon further examination they show irregular outlines that indicate contemporaneous formation. Very similar patches are found in the quartz.

Specimens from some orebodies will show much more chalcopyrite in the primary sulphides than others, although there seems to be no regularity to this variation. It is safe to conclude, however, that an extensive examination of primary sulphide specimens would disclose regional differences in the primary copper content, which in some cases might well account even for the presence or absence of chalcocite.

Surface Ore-Indicators.—The surface of the mineralized area, as a whole, differs from that of the rest of the Burro Mountains region in color and form of outcrops. The limonite and hematite, which have leached out on to the surfaces of the outcrops and rock débris, stain them red-brown, and thereby give the area a generally rusty appearance. Furthermore, the outcrops are more rough and jagged than in the unaffected parts of the district.

Considered in more detail, there are three fairly reliable surface

¹⁵ Simpson, J. F.: Relation of Copper to Pyrite in the Lean Copper Ores of Butte, Mont., *Economic Geology*, vol. iii, No. 7, p. 628 (Oct.-Nov., 1908).

¹⁶ Graton, L. C., and Murdoch, Joseph: *Trans.*, xlv, 42 (1913).

indicators of ore, viz.: (1) the presence of fissures; (2) exceptional rust, either in the outcrops or in the rock débris; and (3) pyrite replacement cavities.

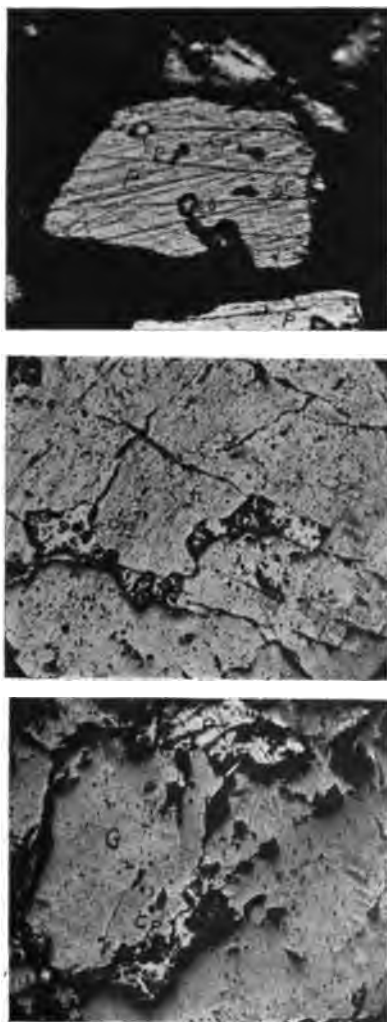


FIG. 22.—CHALCOPYRITE (cp) IN PYRITE (p), FELDSPAR (f), AND QUARTZ. GANGUE (G). REFLECTED LIGHT. $\times 40$.

(1) Fissures.—The ore-bearing fissures outcrop occasionally at the surface, although silicification has been hardly strong enough to enable the majority of them to withstand the weathering agents. When, however, fissures do appear at the surface, they may indicate ore if they are persistent, of sufficient width, and of more than one direction of strike. Without persistence, or a fair width or openness, an occasional

outcropping fissure would hardly indicate a zone of fissuring, and it is the latter in which orebodies are found in the Burro Mountains. The requirement that cross fissures should be present is perhaps open to question, but nevertheless it is doubtful if the solutions which deposited the primary minerals, and the meteoric waters which converted them into ores, were able to work effectively except in zones where a free circulation was promoted by the general breaking up caused by fissures in more than one direction. As noted above, the good orebodies of the district have been found in places where cross fracturing is common. Some ore may be found where cross fracturing is absent, but it will probably not be very plentiful.

In some places, especially about Tyrone, the presence of large fissures is proved, not by the outcrops, but by fragments of vein filling, mixed in with other surface débris, in such locations that it could not have been transported very far.

(2) **Exceptional Rust.**—Especially large amounts of limonite or hematite at the surface show indirectly that mineralization has been more extensive than usual. This indicates ore, provided that chalcopyrite was a constituent of the primary minerals, and that there has been sufficient fracturing to allow surface waters to form an enriched zone. The former could not be proved from any surface evidence, and if chalcopyrite is unevenly distributed, as seems probable from a study of the district, herein lies a difficulty in locating ore from the surface showing. It is not an all-important difficulty, however, because the cases where chalcopyrite seems lacking are the exceptions rather than the rule.

Nearly all the orebodies of the district show exceptional rust in their outcrops. The new Bison orebody is well indicated by a rusted area which is very noticeable from any of the surrounding hills.

(3) **Pyrite Replacement Cavities.**—Cavities which, from their size and shape, must have formerly contained pyrite, are also valuable indicators of ore. They are subject to the same qualifications, however, as given above in reference to the presence of limonite and hematite.

They are to be found above most of the orebodies, and are of special importance over the shaft No. 3 workings, where they seem to be about the only signs that show the location of the ore.

5. *Rock Alteration*

Chloritization.—At a distance from the fractures, beyond the sphere of intense mineralizing action, and where pyrite is very seldom developed, the biotite of the rocks has been changed to chlorite, epidote, and magnetite. This is equally true of monzonite, granite, and quartz porphyry, but results in small quantities of these secondary minerals because of the relative scarcity of original biotite. Other ferro-magnesian minerals

share the same alteration, but they are present in such very small amounts that consideration of them is unnecessary.

As seen in the hand specimen, the biotite thus altered may show merely a greenish color and a lack of elasticity in its plates where the change has not proceeded far, but the final product is a greenish powder, without semblance to micaceous form.

Under the microscope all stages may be observed, from the earlier, where cores of fresh biotite are abundant, through those in which no biotite remains but the outline of the original grain is still clearly defined, to the stages where the chlorite seems to have diffused into neighboring cracks as well as occupying the volume of the primary mica. This last stage produces a very irregular mass of alteration products, which shows only an indistinct resemblance in form to the original biotite.

Rocks which are altered in this manner contain small amounts of sericite.

Sericitization.—In the fractured area, however, sericite is very abundant. It replaces the feldspars in very minute flakes and may possibly develop in quartz in extreme cases.

Plagioclase changes to sericite earlier than orthoclase. The alteration proceeds gradually from the first stage, where a few scattering flakes of sericite appear in the feldspar, to the final product, which is a densely packed aggregate of sericite flakes with occasional grains of the secondary quartz which results from this alteration.

As seen in the hand specimen, the effect is a whitening of the feldspars with the destruction of their luster and cleavage planes. The silky character of the sericite is frequently evident, but it is not general, due to the fineness of the sericite flakes.

Not only has sericite been formed at the expense of the feldspars, but also it has replaced the chlorite of the earlier stage of alteration. In both slides and hand specimens may be observed the large flakes of sericite which occupy the positions once held by the chlorite. The flakes of sericite thus developed are much larger than those formed in the feldspars.

Sericitization is most pronounced immediately adjoining the ore-bearing fractures, and gradually diminishes as the distance from these fissures increases. It is also found intermingled with the quartz-pyrite filling of the fissures, where it has been carried by solutions from the walls.

Sericitization affects all the intrusive rocks of the district. It is perhaps most strikingly developed in some of the fine-grained monzonites which occur as dikes in the northern part of the district. Those phases of this rock which contained very little primary quartz have been converted by this process, combined probably with some kaolinization, into clay-like masses which soften when wet and in many ways resemble

fault gouge. This clay-like material is probably made up of sericite, secondary quartz, and kaolinite, and the intrusive character of its parent rock is proved beyond a doubt by the fact that it often shoots off tongues or branches from the main stringers.

Silicification.—Silicification of the wall rocks has taken place in the Burro Mountains to varying degrees (Figs. 23 and 24). In many places the rock on each side of the fractures is saturated for short distances with secondary silica, which replaces the original minerals, and, at times, may completely obliterate its texture. In some places, however, there is practically no such saturation, the secondary quartz being confined

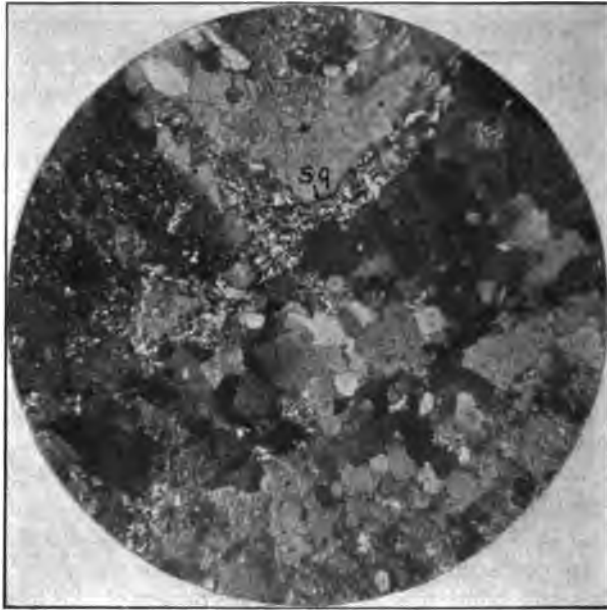


FIG. 23.—VERY FINE SECONDARY QUARTZ (sq) IN MONZONITE.
CROSSED NICOLS. $\times 50$.

to the small fractures, with perhaps an occasional replacement grain. On the whole, silicification has not been nearly as abundant as in many deposits of similar type.

When present in large amounts, secondary silica, with the accompanying iron oxide, gives a strong gossan-like character to the outcrops. These may be noted in any part of the area (Fig. 25). Although these seem rather common in a survey of the district, nevertheless they do not represent any very large proportion of the total number of fissures.

Study with the microscope suggests that there were at least two generations of secondary silica, one of which accompanied the primary sulphides, while the other followed at a later time and filled fractures

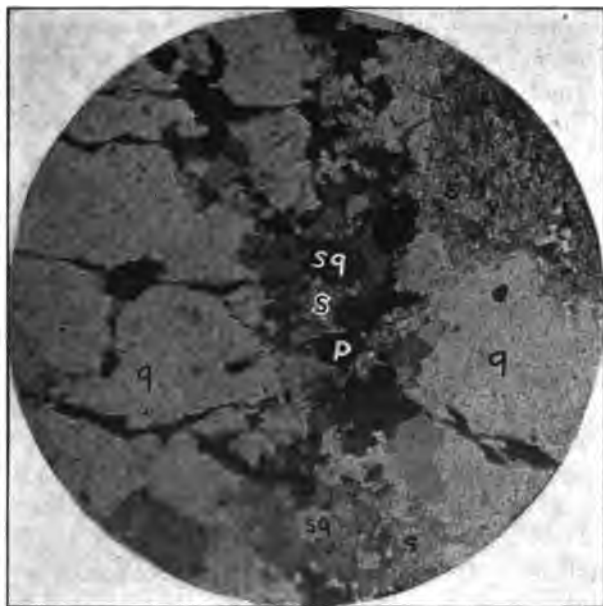


FIG. 24.—VEINLET OF SECONDARY QUARTZ (sq) AND PYRITE (p) IN PRIMARY QUARTZ (q) OF THE GRANITE. SHOWS REPLACEMENT. SERICITE (s) FROM ALTERATION OF FELDSPAR AND ALSO IN VEINLET. CROSSED NICOLS. $\times 50$.



FIG. 25.—ON THE ROAD FROM TYRONE TO SHAFT NO. 2, SHOWING GOSSAN OUTCROPS.

which were developed after the principal period of mineral formation. The latter is sometimes well shown in the hand specimen, as in Fig. 20.

Chloritization, sericitization, and silicification represent the action of hydrothermal solutions. The zonal arrangement of the first two corresponds in a general way with the alteration at Butte, as described by Kirk.¹⁷

Kaolinization.—Meteoric waters have developed near the surface large amounts of kaolinite which give the rock a soft, chalky appearance. As with the secondary metallic minerals, kaolinite extends to varying depths, depending upon the permeability of the rocks.

6. *Genesis of the Ores*

The location of the Burro Mountains deposit on the contact of a post-Cretaceous intrusion of monzonite, its formation after that intrusion and yet previous to a Tertiary volcanic breccia, and a mineralizing activity which is generally considered to-day to be due to hot solutions, point to the monzonite as the prime factor in its existence. In this respect it corresponds with the many deposits of the southwestern part of the United States which are closely connected in origin with monzonite intrusions.

A fractured zone helped to determine the location of the deposit by furnishing channels for the easy circulation of the hot ascending solutions that were given off by the congealing monzonite magma. These hot solutions chloritized the ferro-magnesian minerals and sericitized the feldspars of the country rocks; they deposited quartz and pyrite in the walls of the fissures wherever minor fractures would allow them to penetrate; and they filled the fractures themselves with an aggregate of quartz, pyrite, chalcopyrite, and sphalerite, probably because of a decrease in pressure and temperature as the solutions rose. Chalcopyrite was scarce in the original aggregate, and hence copper very low, but since the fractures were first exposed at the surface by erosion, meteoric waters have been effecting a concentration. This has been brought about by oxidation of the copper to soluble salts, downward percolation, and precipitation as chalcocite upon reaching the reducing environment of the underground water.

Both primary mineralization and downward enrichment were given the most favorable opportunity for their activity where the fracturing was most open, a condition which seems characteristic of the central part of the fractured zone, and hence the best orebodies are found in that portion.

The district is still comparatively undeveloped. Many new orebodies will undoubtedly be disclosed as mining is carried on, and the future of the district seems bright.

¹⁷ Kirk, C. T.: Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii, No. 1, p. 35 (Jan., 1912).

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, November, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Metallurgical Practice in the Witwatersrand District, South Africa

BY F. L. BOSQUI, JOHANNESBURG, TRANSVAAL

(San Francisco Meeting, September, 1915)

INTRODUCTION

THE history of the development of gold metallurgy in South Africa is divisible into two periods: That preceding the introduction of the cyanide process on a commercial scale in 1890; and the 24 years intervening between that important event and the present time.

The period between the discovery of the banket reefs of the Witwatersrand and the year 1890 was one of phenomenally rapid progress in mining development, but of no special interest to the metallurgist. There were no strikingly original or distinctly local advances in the treatment of gold ores. Californian practice in stamp milling and amalgamation prevailed, in mills ranging in equipment between 5 and 50 stamps, operating within a wide range of efficiency, and under the trying conditions usually encountered in a new and isolated field. The somewhat incomplete records of gold output of that period show that in December, 1887, there were 20 producing companies in the Witwatersrand district, of which 10 had mills, the largest being of 20 stamps. The only official record of returns for May, 1887, was that of the Wemmer mine, whose modest achievement with a 5-stamp mill was 100 tons crushed, producing 887 oz. 3 dwt. of gold. This is interesting in view of the present-day output of this field, which in 1913 reached 8,430,998 oz. The rapid increase of production during this first period (May, 1887, to January, 1891) is shown by the following summary for the year 1890:¹

Total tons crushed	702,828
Number of stamps in operation	1,046
Average number of tons crushed per stamp per day	2.36
Yield of gold, ounces	494,817
Yield per ton, pennyweights	13.36

The stamp duty ranged between $1\frac{1}{2}$ and 4 tons per day, according to weight of stamp, fineness of screen, and operating skill available. The prevailing weight of stamp was about 900 lb., and the screen, 900 meshes to the square inch.

¹ *Annual Report of Johannesburg Chamber of Mines, 1889-90.*

Assuming that the recovery at that time was as high as 60 per cent., the recoverable gold lost in tailing from the ore treated during this first period, in spite of the precaution taken in some instances to impound this product, must have been very considerable. It is evident, therefore, in view of the colossal strides made by the industry at this stage, and the promise revealed for the future, that the advent of the cyanide process was most opportune.

The introduction of the new process was the direct result of trials conducted at the old Salisbury battery in 1890 by the Cassel Gold Extraction Co. of Glasgow, which controlled the MacArthur and Forrest patent rights. Its first application on a commercial scale was in the same year, in the treatment, by contract with the Gold Recovery Syndicate, of 10,000 tons of tailing on the Robinson Gold Mining Co.'s property, under the direction of G. A. Darling. In February, 1891, W. A. Caldecott started the cyanide works at the Sheba mine in the Barberton district; and a year later, Charles Butters erected a large plant at the Robinson mine to treat all the tailing from that property. From this time cyaniding was recognized as an indispensable adjunct to every mill. It was not, however, until 1894 that the direct treatment of battery slime became a success. The first decantation slime plant was erected in this year at the Crown Reef mine, under the direction of J. R. Williams, at that time chief metallurgist for the Rand Mines, Ltd., who evolved the process after several years of practical trials.

In the meantime, in 1894, the Siemens-Halske electrolytic method of gold precipitation was introduced at the Worcester mine, and was adopted by several other properties. But in spite of able advocates it never won general acceptance, owing, broadly, to the extreme delicacy of the process, the formation of troublesome by-products, and the high degree of skill required to maintain uniform efficiency; and after exhaustive comparative trials with zinc and electrolytic deposition at the Nourse Deep mine in 1898, it was finally abandoned in favor of zinc. Fortunately, a serious objection to zinc precipitation—namely, the difficulty of precipitating from the very dilute cyanide solutions used in slime treatment—was partly overcome at about this time by the use of acetate of lead as a "dip" for zinc shavings, which coated the zinc with metallic lead, forming a zinc-lead couple which greatly promoted galvanic action in the zinc boxes.

With the reversion to the use of zinc, Rand metallurgical practice may be said to have crystallized into a general scheme of milling and of separate sand and slime treatment by cyanide, which came to be looked upon as typically South African, having been, with the exception of stamp milling, evolved by local chemists and metallurgists. Even the introduction of tube mills in 1904 did not seriously modify the main features of the system. The tube mill became simply an accessory to the stamp. This survival of the broad principles of ore treatment, as developed by the

clever metallurgists whose services the industry was fortunate enough to secure at the beginning, has been attributed to a narrow disinclination to adopt new methods as applied in other parts of the world. This view, however, must be very considerably modified.

It is true that the metallurgists of the Rand were slow to accept the two most notable appliances introduced into the treatment of gold ores in the last 10 years—the tube mill and the vacuum filter. The former was introduced in Africa in 1904, after having been for some time a success in Australia; the latter in 1909, after four years of brilliant success in America and elsewhere. The Boer war undoubtedly retarded the introduction of tube milling, and it must be admitted that when once tried, and their merits proved, both innovations were very generally and enthusiastically adopted. The fact that such radical measures as “all sliming,” dry crushing, and numerous schemes for continuous treatment of battery pulp by agitation have not found favor on the Rand is evidence, not of lack of enterprise or open-mindedness, but of the proper estimation of unique local conditions, and the conviction that any system of treating the low-grade Rand ores must fulfill the essential requirements of “foolproof” simplicity in operation, low maintenance and treatment cost, and high efficiency.

The local peculiarities of the district have not always been understood by critics. The ore of the Rand is not only low grade (ranging in gold content between $4\frac{1}{2}$ and 9 dwt. per ton), but is simple in composition, containing as high as 70 per cent. of its gold in amalgamable form, and a small portion in pyrite, which is quite amenable to cyanide treatment. Obviously, an ore of this grade and character does not permit of elaborate appliances or refinements of treatment. Moreover, it must be remembered that unskilled labor is cheap on the Rand, while skilled labor and technical advice are expensive. Supplies are also high priced; and electric power is comparatively costly. The country, being flat or gently undulating, does not afford such favorable sites for reduction plants as where gravity can be taken advantage of in the flow of mill pulp. This disadvantage of site also means added expense for disposal of residue, which must be carefully impounded or stacked. It will be seen from the foregoing that the economic aspect of Rand metallurgy is a study of some complexity, involving a nicety of balance between technical and financial considerations, and requiring the utmost caution on the part of metallurgical and mechanical advisers. This will explain the retention of methods and apparatus which, to the stranger fresh from the exhilarating atmosphere of innovation, may seem crude and awkward, but which have survived the test of competition with rival processes, and have proved to be the best available for our special purpose. When the very high gold extraction on the Rand (94 to 97 per cent.) and the low reduction cost (3s. and less per ton in the newer plants) are considered, the scope for economic improve-

ments does not seem great. It may be mentioned here, however, that though, broadly speaking, the same general system of reduction is in use throughout the Rand, there are many variations in detail between old and new plants, both in mechanical arrangement and treatment, which will be dealt with in their proper place, under the following subdivisions:

Step in Treatment	Variations
I. <i>Ore Sorting and Breaking</i>	(a) Ore washed in trommels or in stationary chutes. (b) Ore sorted on slowly moving belts or on revolving tables. (c) Ore broken in two types of crusher, those in common use being the Blake swinging jaw type and the Gates gyratory.
II. <i>Stamp Milling</i>	(a) A range in weight of stamps from 1,150 to 2,000 lb. (b) Considerable variation in details of battery construction; also variation in mesh of battery screen dependent chiefly upon tube-mill facilities.
III. <i>Amalgamation</i>	(a) Use of battery plates, and so-called tube-mill plates for secondary amalgamation after tube milling. (b) No battery plates; all amalgamation done after tube milling. (c) Tube-mill plates, shaking and stationary. (d) Considerable variation in ratio of area of plates to tonnage.
IV. <i>Tube Milling</i>	(a) Tube mills of varying dimensions. (b) Varying ratio of number of tube mills to stamps. (c) Devices for thickening pulp and feeding tube mills. (d) Kinds of "liner" in use.
V. <i>Classification of Pulp</i>	(a) Coarse sand separated out and sent to tube mills. (b) Separation of sand from slime. (c) Classification in spitzkasten or hydraulic cones.
VI. <i>Treatment of Sand</i>	
1. <i>Collecting</i>	(a) Collecting tanks superimposed on treatment tanks, or on ground level, and fed by (1) movable hose, (2) Butters and Mein distributors. (b) Collecting tanks filled direct from classifiers or with solution-borne pulp from Caldecott sand-filter tables.
2. <i>Treatment</i>	(a) Sand dropped from superimposed collectors into treatment tanks. (b) Sand transferred by belt conveyor from collectors on ground level. (c) Sand collected and treated in the same tank from Caldecott sand-filter tables. (d) Sand residue trammed to dump in trucks. (e) Sand sent to dump by aerial bucket conveyor.
VII. <i>Treatment of Slime</i>	(a) By circulation with pumps and decantation. (b) By agitation in Pachuca tanks, followed by filtration with Butters vacuum filter.
VIII. <i>Gold Precipitation</i>	(a) On zinc shavings. (b) On zinc dust in Merrill presses.
IX. <i>Smelling Methods</i>	(a) In Tavener lead furnaces, with cupellation. (b) Direct smelting in pots.

SORTING AND BREAKING ORE

The term "banket" is the Dutch word for almond rock, or almond "candy," as we should say in America; and was applied by the Boers to the conglomerate ore of the district which, with its white pebbles incased in the darker matrix, bears a resemblance to that sweetmeat. The object of sorting, which was first introduced in 1892, is to separate this banket ore from the comparatively barren quartzites, or waste. The care with which sorting is done and the proportion of ore rejected as waste (from 14 to 20 per cent.) vary at the different mines, according to facilities, occurrence of orebodies, and a number of economic considerations.

Ore sorting on the Rand has been the subject of a good deal of discussion. It has been suggested that all sorting should be done underground, while authorities have differed as to what proportion should be done underground and what on the surface. A kind of crude preliminary sorting is now done below the surface, consisting in hand breaking of pieces of ore too large for convenient handling. Obviously, the technique of ore sorting is a matter to be dealt with independently on each mine. The present practice is to sort above ground, at a central sorting and breaking station, where both operations are usually carried on under the same roof. These stations are of great variety, but the general principle is the same in all.

The ore, on reaching the surface, is divided into two products by means of grizzlies, which take it direct from the skips in the headgear; the "fines," which it is impossible to sort economically, go direct to the mill bin; the coarser product is then delivered either (1) to grizzlies or (2) to trommels, where it is washed and screened preparatory to hand sorting. Or, in older plants, the grizzly and trommel are omitted, and washing is done by means of sprays on the same belt or table on which sorting is performed. Washing the ore is necessary in order to enable sorters to distinguish between banket and waste. It is now recognized that this operation cannot be thoroughly performed except in washing and screening trommels, and the latest plants are all equipped with this device. From the trommel or grizzly the washed ore is delivered to either (1) a revolving table or (2) a traveling belt. The former is annular in shape, from 18 to 30 ft. outside diameter, the horizontal rim or working surface not exceeding 42 in. in width. About 3 ft. of space is allowed for each sorter—young native lads being chiefly employed. The ore makes nearly a complete circuit and is finally removed by a plow, whence it passes to the breakers. The sorted waste is either dropped below into a bin, or on to the lower deck of a double-decked table, whence it is removed by a plow, and thence to the waste dump.

The advantage of the sorting table is that it permits of a compact plant, and so simplifies supervision; but the belt has now almost entirely

superseded the table, its advantages being that it is cheaper to install and operate and that it elevates and conveys the ore as well as provides a sorting surface. These belts vary in width between 30 and 40 in., operate at an angle of about 10° , and are provided with a special form of idler to give a concave surface for draining and preventing spill. They are made either of canvas, or of canvas and rubber. Their capacity is between 50 and 100 tons per hour, according to width.

The waste from the sorting belt is dropped either into (1) a bin, for direct transfer to dump, or (2) on to waste belts, arranged in a variety of ways. Or the returning portion of the continuous sorting belt may be made to deliver the waste into trucks, bins, or a cross belt to dump. A considerable problem in sorting stations is the disposal of the "washings" or muddy water carrying grit and slime which flows from trommels, grizzlies, or the sorting belt, according to the method of washing adopted. The following methods are in common use: (1) these washings are settled in special tanks or pits, and periodically shoveled out and mixed with the ore going to the mill bins; (2) they are delivered direct to centrifugal pumps, and discharged into the mill-pulp stream; (3) or they are subjected to screening in secondary trommels, the washed oversize passing to a fines belt, which joins the main conveying system, and so direct to the mill bins, the escaping water and fine material going to centrifugal pumps and thence to the mill-pulp stream.

The imperfect arrangements for washing ore and for disposal of these washings in many sorting stations on the Rand, and the crowded, patched-up appearance of the majority of these stations, lead to the conclusion that this stage in the metallurgy of Rand ores has been rather neglected in the past. One wonders why this step in reduction should not have crystallized long ago into some good, generally accepted scheme—instead of which we find the greatest variety of ill-designed, crowded, messy plants, seemingly in the first crude stage of development. It is only in a few of the newest mills that the design of sorting and breaking stations has received sufficient attention. We find these well lighted and spacious; the usual spill and mess prevented by the use of proper washing trommels, and neat arrangements provided for the collection of washings and their direction into suitable pump pits.

In the next plant to be built by the Rand Mines, Ltd., it is expected that the sorting and breaking station will embody all the latest improvements and facilities (Fig. 1). The bad practice of putting fines and slimy ore in the mill bins will be obviated. By means of the secondary washing trommels only the fines that will not pass a screen of 9 holes to the square inch will be sent to the mill bins; all finer material will be diverted to the tube mills. These washings should properly not join the mill stream direct, for the reason that, as the crusher station runs only 8 hr. in the 24, this large accession of water during this period presents a serious

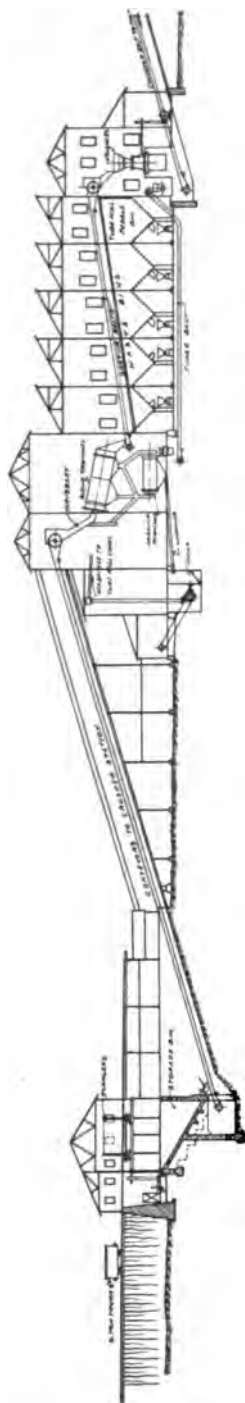


FIG. 1.—ELEVATION OF BREAKING AND SORTING STATION, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

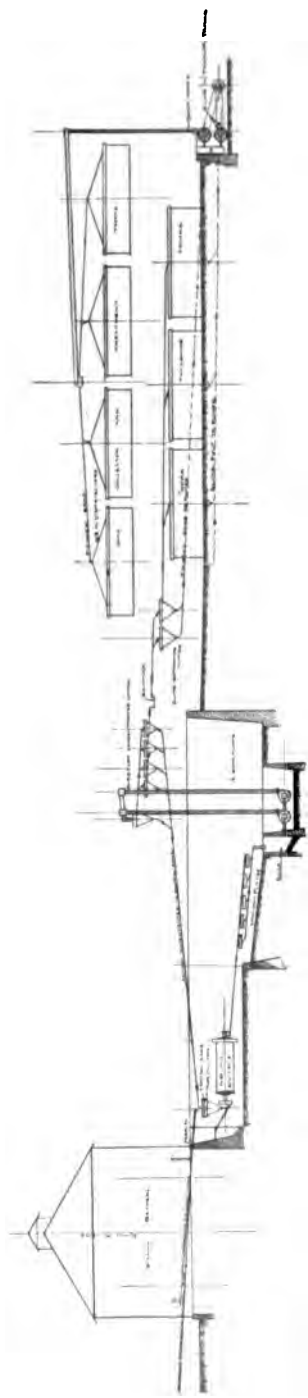


FIG. 5.—DIAGRAM OF CLASSIFICATION SCHEME, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

detriment to uniform classification. It is important, therefore, that it should be specially treated before reaching the mill. For this new mill a continuously operating special plant has been devised to deal with this troublesome product. It will run into suitable classifiers, the underflow of which will go to the tube mills; the bulk of the water and slime will be diverted to settlers, the last of the series representing the reservoir supplying the washing trommels. Thus the mill will be protected from irregular and large accessions of water, so undesirable where proper classification is recognized as of vital importance.

The sorted ore is delivered to breakers, of which there are two types in common use, the Blake-Marsden jaw type with its variations, and the gyratory type, of which the Gates is generally used. The gyratory breaker is the type now accepted by the Rand Mines and Consolidated Goldfields groups, and is considered more efficient and cheaper to operate where large tonnages are handled; the jaw crusher, however, is a simpler machine, of relatively less weight and requires less vertical height. The gyratory crusher is particularly well adapted to the finer breaking, while for coarse preliminary breaking, before sorting, where the production of fines is objectionable, the jaw type is preferred.

There is still a divergence of opinion as to the extent to which breaking before sorting should be carried. In some mines breakers are placed underground, with the jaws set to 4 to 8 in. opening, to reduce the ore to suitable size for handling in the skips and for sorting; more often these breakers adjoin the headgear. Where they are omitted altogether, it is in recognition of the theory that it is better to send a high percentage of waste to the mill than reduce the waste to smaller pieces, and so improve the accuracy of sorting while increasing the aggregate gold content of the waste dump. It will be seen from the foregoing that the practice in dealing with the ore before it goes to the mill is exceedingly variable. Within the last four or five years, however, the technique of sorting and breaking has greatly improved, and there is now apparent, among the groups erecting new plants, a disposition to collaborate in an effort to bring about more uniformity in practice and mechanical arrangement.

STAMP MILLING

The gravitation stamp has retained its place on the Rand as the simplest and most effective appliance for crushing ore. Until the recent introduction of the heavy single-stamp unit, the general features of ore crushing have undergone no very radical change since the introduction of Californian mill practice over 20 years ago. The most marked departure from the typical 900-lb. stamp of those days is in the progressive increase of weight. This interesting evolution from the light to the heavy stamp has probably now reached its limit in the adoption of 2,000-lb. stamps in

mills erected within the last few years. Changes in detail of battery construction have been necessitated by this progressive change in weight; exigencies of high stamp duty, requiring larger bin capacity, have also brought about modifications in constructional detail.

It would be impossible in the scope of this paper to deal with the many variations in stamp-battery construction to be found on the Rand. I shall consider only a few of the more important stages in progress, which have led to the introduction from America of the single-stamp unit, or Nissen stamp.

The first step in advance was the adoption by the Robinson Gold Mining Co. and others of 1,250-lb. stamps. This was the maximum until 1907, when a distinct impetus was given to a consideration of heavy stamps by the results of a series of trials conducted by the Consolidated Goldfields group under the direction of W. A. Caldecott. It was obviously desirable to ascertain the highest duty attainable from as heavy a dropping weight as could be sustained by an improved type of mortar box and concrete mortar block. It was apparent that any increase of duty would mean a corresponding decrease in number of units necessary, and, in consequence, lower initial and operating cost.

The Goldfields trials led to the installation of three large batteries of 1,550-lb. stamps, and more recent mills have adopted heavier weights up to 2,000 lb. Gradually timber-pile foundations were abandoned in favor of reinforced concrete, with or without the interposed anvil block. The stem and cam-shaft breakage attributed to this cast-iron base, and the higher cost of batteries using the anvil block, led to its abandonment in most of the more recent mills, although it still has its advocates. The accepted practice is now to bolt the mortar box direct to the concrete block by means of renewable bolts so arranged that they can be tightened while the battery is in operation. A cushion of hair felt or of sheet rubber, about $\frac{3}{8}$ in. thick, is placed under the mortar box.

A notable departure from the standard type of battery construction was the case of the City Deep, Ltd., which in 1909 adopted reinforced concrete piers instead of king posts, and separate bearings between the cams, for a new battery of 2,000-lb. stamps. This design was intended to obviate cam-shaft breakage by affording greater bearing surface, and to insure better alignment, in view of the greater strain imposed upon the cam shafts by the heavy stamps. At the City Deep the concrete piers built up from the 10-stamp block support a single rigid steel casting, into which all the various parts of the usual battery superstructure are consolidated. The main feature of this casting is the heavy fish-bellied girders, with eight upward projections or fingers, terminating in cup-shaped bearings for the cam shaft. Each 10-stamp shaft is thus given 11 bearings instead of the three usually provided. Theoretically, the innovation was sound, and ingeniously worked out. In practice, how-

ever, complications developed. It was found that a true alignment of bearing surface was practically impossible to maintain, without most troublesome and constant adjustment of the bearings themselves. Moreover, as the advantages of such a design were obviously dependent upon the integrity of the single ponderous casting, it was expected that this rigid frame would resist the enormous strain of continuous vibration. Unfortunately this casting in the course of time weakened, and developed fractures, requiring reinforcement and patching.

Following the City Deep installation, the old standard design of battery was reverted to and has been, with one exception, retained. In mills erected within the last five years we find the weight of stamps ranging between 1,550 lb., which some authorities reckon to be the conservative limit for combination of five units in a single mortar box, and 2,000 lb., which many regard as excessive weight, from the point of view of efficiency, convenience, and cost of maintenance.

The introduction of the tube mill in 1904 had much to do with the development of heavy stamps. Until that time fine crushing in the battery was necessary, in order to secure good recovery by amalgamation and cyaniding; but with the advent of the tube mill it became possible to use screens of coarser mesh, the limiting factor being the coarseness permissible for good amalgamation. It was obvious, therefore, that if battery amalgamation plates could be dispensed with, the only factor limiting coarse crushing would be the size of particles permissible in the tube-mill circuit. The removal of battery plates was therefore the next step toward attaining maximum stamp duties; and now in all the newest mills amalgamation is only carried on after tube milling. We thus find the function and scope of the stamp quite altered; it is now recognized as a crushing device pure and simple, hampered in capacity by tube-mill limitations only; so that, beginning with duties of a few tons from 900-lb. stamps when screens of 900 mesh were necessary, we are now obtaining duties of approximately 30 tons, with 9-mesh screens (0.27 in. aperture) from 2,000-lb. stamps. There is still, however, a considerable range in weight and duty of stamps on the Rand. Many older plants still retain battery plates, and are not provided with the same tube-mill facilities as newer mills. This is because capital expenditure on old plants is not always warranted by the life of the mine and the financial position of the company; this must be borne in mind, as otherwise the variable data given in the following tabulation, from the mills controlled by the Rand Mines, Ltd., may appear inexplicable.

Mills on the properties of other groups are obtaining stamp duties as high as 20 tons per day, from 2,000-lb. stamps, with 9-mesh and 4-mesh screens; but the highest duty yet recorded is that of the 2,000-lb. Nissen stamps at Modderfontein B., 29 tons through 9-mesh screen.

Averages for the Year 1913, Rand Mines, Ltd.

Company	No. of Stamps		No. of Tubes		Average Dropping Weight of Stamp	Duty of Stamp per 24 hr.	Screens Used
	Erected	In Operation	Erected	In Operation	Pounds	Tons	Mesh
Modderfontein B.*	80	77	5	5	1,650	14.1	12
New Modderfontein	180	143	7	6	1,320	9.8	67
Rose Deep	300	261	7	7	1,250	7.9	122
Geldenhuis Deep	300	242	7	5	1,133	6.9	231
New Heriot	70	54	2	1	1,250	6.7	375
Nourse Mines	260	193	7	5	1,268	8.2	183
City Deep	194	106	9	5	1,600	11.8	77
City and Suburban	160	137	2	2	1,200	6.0	440
Village M. Reef	220	168	6	3	1,105	7.0	400
Village Deep	180	164	7	5	1,312	8.8	248
Ferreira Deep	280	233	7	6	1,137	7.5	258
Robinson	250	229	6	6	1,230	7.9	165
Crown Mines A.	300	246	10	8	1,262	9.6	92
Crown Mines B.	200	173	6	4	1,212	9.0	106
Crown Mines C.	160	129	9	8	1,672	15.3	11
Bantjes	100	79	3	3	1,415	10.0	62
Durban Roodepoort Deep	100	91	3	3	1,500	8.6	148

* The 16 Nissen stamps were not erected until 1914, making a total of 96.

This brings me to the subject of the Nissen or single-unit stamp, 16 of which have just been installed at the Modderfontein B. mill as a result of most favorable trials conducted at the City Deep in 1911.² These trials incontestably demonstrated certain points of superiority of this type of stamp, showing, for example, that pound for pound of dropping weight, the Nissen stamp crushed about 30 per cent. more rock per day and had an increased efficiency of 35 per cent. over the ordinary stamp of equal weight. This stamp has been well described in Mr. Nissen's paper, cited above.

The better results obtained appeared to be due to the elimination of the admittedly weak mechanical features of the older design. The serious defect of the conventional 5-stamp arrangement is that it is impossible to adjust and distribute the feed properly. It is obvious that in the multiple box all the stamps are not doing the same amount of useful work. The complex and mutually opposing currents set up within the mortar box make uniformity of feed impossible. The uncrushed ore is not distributed evenly over the dies, so that coarser pieces are not always most suitably arranged under the falling stamp. This difficulty in controlling feed is undoubtedly largely responsible for broken stems and shoe necks, and the uneven wear on dies. Another serious defect of the multiple principle is that with the long screen and the turmoil of pulp set up within

² *Transactions of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xi, p. 110.

the box, there is no *positive* discharge of the crushed ore when reduced to the desired size, and only that material adjacent to the screen can be discharged. Another obvious mechanical defect of the 5-stamp principle is that different sections of the mortar casting are subjected to continuous and rapid blows, which produce a rocking tendency eventually loosening foundation bolts. These, if kept tight to insure rigidity, will stretch and eventually break.

The distinct advantages of the single-stamp principle may be summed up briefly as follows:

1. Owing to the mortar being circular, the screen can be extended around the stamp for the greater part of the circumference of the mortar, so that it is equally distant from the stamp throughout its full length. The screen is therefore always at right angles to the direct splash of the pulp, in the most advantageous position.

2. At each blow of the stamp the pulp is forced radially against the screen, so that all particles sufficiently reduced are discharged. Owing to the mortar box being circular, each time the stamp is raised all the material in the mortar flushes to the center, to be struck by the falling stamp. It follows, therefore, that the uncrushed ore is automatically returned to the crushing zone, so that the best conditions of feed are maintained.

3. The blow being always received in the vertical axis of the box, the mortar remains rigid on its foundation. This is noticeable in the case of the Nissen stamps at the City Deep, where, after three years of use, the box is apparently as rigid on its base as on the day it was fixed in position.

4. One of the important features of the Nissen box is that it can be cast with a minimum likelihood of shrinkage strains, which are so destructive to ordinary 5-stamp mortars. The foundation bolts give no trouble, not being subject to undue strains.

5. Another advantage of the unit principle is the more continuous operation and flexibility of the entire plant, as each stamp can be put out of commission independently.

6. An interesting feature of this type of stamp is the comparatively even and flat wear of the dies, which results from the return wash of the ore to the center of the mortar with each stroke of the stamp, and from the increased number of rotations of the stamp due to wider cam and tappet faces.

The only objection seriously urged against the single-stamp unit in South Africa is the initial cost of erection, as compared with the Californian. On the assumption that the single-stamp installation required a longer mill building than a Californian for a corresponding tonnage, we should require longer bins, greater fall for launders, increased depth of pump pit and higher pump lift. But this objection is practically nullified by experience at Modderfontein B., where tonnages of 14.25 and 29 tons from Californian 1,650-lb. stamps and Nissen 2,000-lb. stamps,

respectively, both using 0.27-in. aperture screen (9-mesh), indicate a negligible difference in length of mill required. It is true that 2,000-lb.

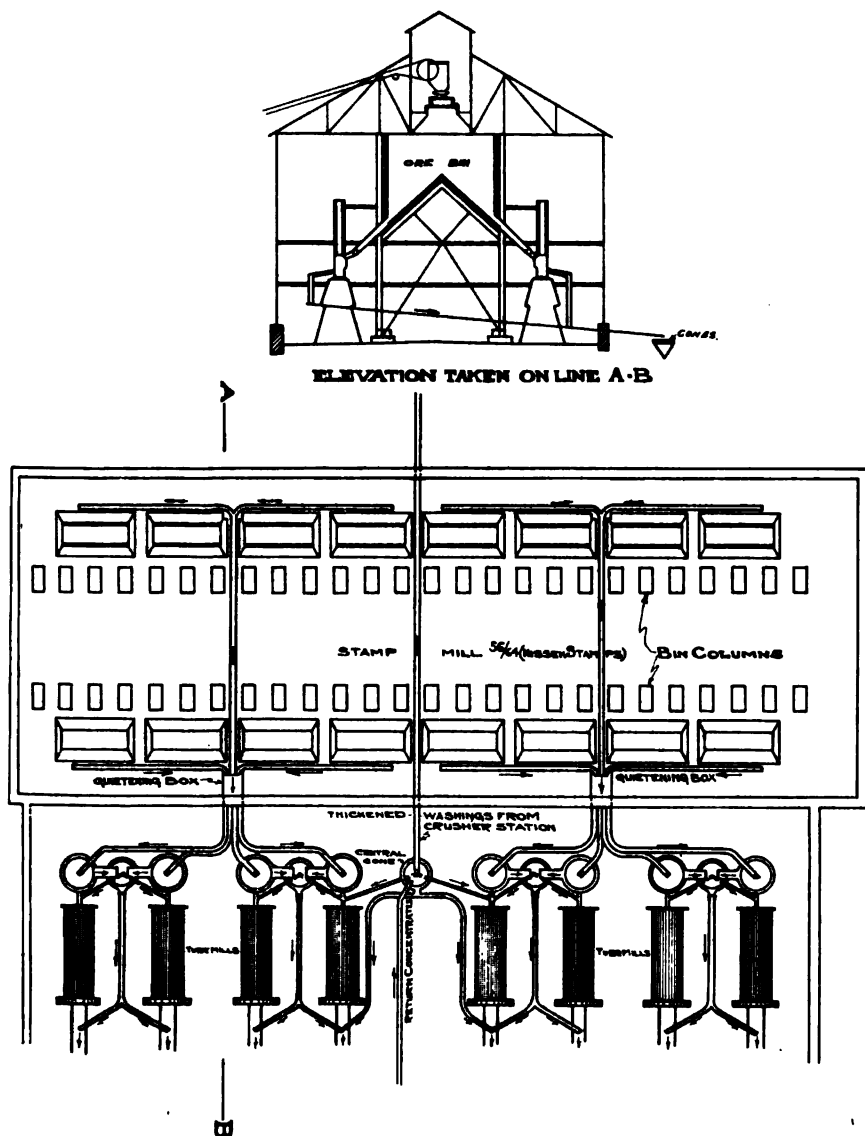


FIG. 2.—FLOW PLAN OF STAMP MILL, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

stamps in multiple arrangement have recorded duties as high as 20 tons; but assuming that 1,650 lb. is the economic mechanical limit for stamps arranged in series of five, the chief objection to the Nissen stamp may be

regarded as unimportant. On the basis of the above data, 50 1,650-lb. Californian stamps would crush 713 tons per day; the same tonnage can be crushed by 24 Nissen stamps. The former would require a mill building 117 ft. long, if arranged in single line; the latter, 133 ft. Even if we compare the room taken up by the two types of stamp of equal weight, the disadvantage of increased length of building for single stamps in this case would be easily offset by improved efficiency.

The floor plan of the proposed new works of the Rand Mines, Ltd., is shown in Fig. 2.

AMALGAMATION

The recovery of gold on amalgamated plates is still a very important department in Rand reduction plants, in spite of its gradual restriction in some cases to a relatively small amalgamating area, and the tentative proposals made from time to time to eliminate it altogether. It would simplify metallurgy enormously, both from a technical and metallurgical point of view, if the cyanide process could be depended upon to extract all the gold from Rand ores. In America, in recent installations, amalgamation is omitted. In these cases, however, the gold is found either (1) so finely divided, though free, that its dissolution by cyanide is sufficiently rapid and complete, or (2) it does not exist in amalgamable form. On the Rand, where we find recoveries by amalgamation as high as 70 per cent., the greater portion of the coarse gold, variable in quantity at different mines, is only economically recoverable on plates; and there is, therefore, no likelihood of this step in treatment ever being dispensed with.

Many factors have influenced and changed amalgamation practice on the Rand. In the early days "inside amalgamation" was carried on in the spacious and specially designed mortar boxes provided with plates; but high mercury losses, the impossibility of obtaining a reliable screen sample, the inconvenience of cleaning up, and the necessity of modifying the design of mortar box to a narrower and more efficient type for crushing purposes, were the causes of the gradual abandonment of this practice. Apron battery plates are still retained in many mills, one to each battery of 5 stamps, and are of the conventional type, the average size being 5 by 12 ft., inclined to a maximum of about $1\frac{1}{2}$ in. to the foot. When tube milling was introduced, making coarser crushing possible, supplementary plates were placed after the tube mills, to offset the reduced recovery from the battery plates, and to recover the gold released in the tube mills. A shaking movement was imparted to these plates, similar to that of a concentrating table, by means of hard-wood springs and an eccentric driven from a countershaft. As many as seven of these plates, 5 by 12 ft., were placed below each tube mill, set at a grade of about 10 per cent. The shaking movement was considered necessary to insure a good distribution and flow of pulp; but as tube mills became more effi-

cient and numerous it was found that a stationary table, set at 18 per cent. grade, was quite as effective; and in many mills the tube-mill plates were changed from shaking to stationary.

The practice at present (Fig. 3) is as follows: Amalgamation with and without battery plates; classification of stamp-mill pulp in tube-mill cones, the underflow of these going to tube mills and thence over tube-mill plates; the overflow going either (1) direct to the cyanide plant, (2) to a separate set of amalgamating plates, or (3) to join the main pulp stream leaving the tube mills. It will be seen that in (1) and (2) only the tube-milled pulp is amalgamated on the secondary plates, while in case (3) all the pulp passes over these plates.

This difference in practice is affected by a number of considerations. We find the Consolidated Goldfields using only three stationary plates to a tube mill, after re-classifying the overflow from the tube-mill cones and bypassing a considerable amount of pulp without secondary amalgamation; while the Rand Mines, in their newer plants, use six tube-mill plates, and bypass little or no pulp. Undoubtedly the former method is the simpler; there is less capital expenditure for plant and buildings, less capital held over in the form of amalgam, and reduced operating expense. But experience on the Rand would seem to show that it is impracticable to fix any definite limit for plate area. By restricting this area to narrow limits an increased burden of extraction is thrown on the cyanide plant, involving an irregular realization from a considerable portion of the total gold extracted, which is undesirable where the maintenance of a uniform and easily available yield is of so much importance from the point of view of mine administration. But apart from this consideration, variations in ore, as regards grade, coarse-gold content, and general composition, certainly affect amalgamation; and as these factors cannot be depended upon to remain constant from month to month even on the same mine, I am of the opinion that the radical reduction in plate area recently advocated can be regarded as safely applicable only in special cases. The amalgamating facilities at Modderfontein B. may be considered a mean between the two extremes; here there are no battery plates, but 30 stationary 5 by 12 ft. tube-mill plates, or a total of 1,800 sq. ft., with capacity of 39 tons of ore per plate per day. Trials have been made at this plant with a view to reducing the number of plates, but the appearance of free gold in the sand residue during these trials, and during periods when the 30 plates were overcrowded by a sudden increase of tonnage, led to the conclusion that, in this instance at least, any reduction of plate area would be inadvisable.

The details of amalgamation do not differ essentially from practice in other parts of the world, which has been described in a number of text books. The plates are periodically "dressed" by brushing up the accumulations of concentrate, or "black sand," with adherent amalgam,

followed by the usual sprinkling with mercury, and vigorous rubbing to produce the necessary uniformity and softness of surface. The operation of "scraping" is carried out once a day, usually in the morning, by means of steel scrapers, which remove the bulk of the gold accumulated during the previous 24 hr. To prevent an excessive accumulation of gold on plates, "steaming" is resorted to at varying intervals, the usual practice being to steam one-third of the plates every month, so that all plates obtain this treatment about four times a year. The procedure is to place a tight wooden cover over the plate and introduce steam until the amalgam is sufficiently softened by the heat. The plate is then scraped until only enough amalgam remains to insure a good surface. All accumulations of amalgam and "black sand" are treated either in a grinding pan or amalgam barrel, the latter being preferred. Grinding with steel balls is carried on for about 2 hr. The contents of the barrel are then run slowly off into a *batea*, where further amalgamation takes place, the residual slimed black sand overflowing to suitable boxes, where it is retained for further treatment. All amalgam from mercury traps, launders, etc., is similarly treated. The resultant product from the grinding barrel and *batea*, when the bulk of the gold has been separated from it in the first operation, is subsequently treated in a small tube mill, whence the outflow is allowed to pass over an amalgamated plate. To recover the major part of the residual gold not saved in these two operations, the slime thus produced is treated either in separate tanks by prolonged leaching, or in a small air agitating vat, followed by decantation. By this means extractions of over 95 per cent. are obtained from this product. The tailing is sometimes discharged into the slime plant and mixed with a current slime charge.

TUBE MILLING

The determination of the exact scope of the tube mill in crushing ore, and the conditions under which it would work most effectively, was not arrived at until some time after its introduction. The application of the diaphragm cone, which made possible a uniform feed of easily controllable moisture, and the introduction of heavy stamps, which permitted coarse crushing within the wide range necessary for fixing, by trial, the economic scope of the tube mill, were important factors in the development of this important auxiliary to crushing, whose value and limitations are now pretty well understood.

The obvious desideratum in distributing the work of crushing was to ascertain the economic point of separation between stamp milling and tube milling in the production of a final product sufficiently fine to yield the maximum net profit. To determine this point of economic balance has been no simple undertaking, but has entailed a vast amount of practical investigation.

The cost of tube milling had an important bearing on this problem. The first step toward reducing operating expense was in the substitution of selected pieces of banket ore for grinding purposes, in place of the imported pebble. Then it became apparent that a tube mill worked more efficiently on coarse than on fine pulp; and the opinions of metallurgists finally converged to the now generally accepted view that the product of a 9-mesh battery screen (0.272 in. aperture) is about the economic limit of size for tube-mill feed on the Rand, in producing a final product most suitable for cyaniding. Beyond this point, except in special cases, the tube mill encroaches on the domain of the stamp.

This point having been satisfactorily settled, it remained to determine the proper ratio of tube mills to stamps. At present this ratio is extremely variable, dependent upon many factors. But in new mills, the usual allowance is based upon the result of extensive trials in which all the leading groups have participated. In recent practice, the tendency has been to increase the ratio of tubes to one 22 ft. 6 in. by 5 ft. 6 in. tube mill to ten 2,000-lb. stamps, or one tube mill to 200 to 250 tons per day of 9-mesh product. At Modderfontein B. the ratio, when the sixth tube mill shall have been erected, will be one of the latter to 264 tons per day of 9-mesh battery product. At the proposed new mill the ratio will be one tube mill to seven Nissen stamps, or one mill to 203 tons of 9-mesh product per day. Latter-day practice aims at the production of certain screen grades in the various stages of reduction, which will give the most suitable final products for sand and slime treatment. The following gradings are fairly representative of the work being done at a modern plant using efficient classification and vacuum filtration:

	+ 60 (0.01 in.)	+ 90	- 90 (0.006 in.)	- 200 (0.0025 in.)
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Entering tube mill:				
Main circuit.....	85.81	8.08	6.11	
Concentrate return.....	58.30	30.87	10.83	
Leaving tube mill:				
Main circuit.....	18.74	23.58	57.68	
Concentrate return.....	10.59	31.52	57.89	
Final pulp before slime separation.....	1.40	13.87	84.73	
Sand (39 per cent. of total ore).....	9.28	38.76	40.94	11.02
Slime (61 per cent. of total ore).....			10.00	90.00

(Over the period represented by above gradings, the extraction by cyanide was 93.7 per cent.)

We now come to a consideration of the more important improvements in the details of tube milling, as locally evolved. In reviewing the progress of this branch of reduction since the general adoption of tube

mills by the gold mines of the Rand in 1904 to 1905, it must be confessed that divergence from the practice and design followed in the initial stages has been significant. Up to a few years ago the 22 ft. by 5 ft. 6 in. tube mill, standardized on these fields, remained almost unchallenged; and in the opinion of many, the change since introduced to greater diameter and less length has still to be justified by practical comparative tests. Modifications in methods of introducing the pulp, quantity of feed, percentage of moisture, crushing load, speed of rotation, and in methods of lining have naturally resulted as experience matured, and have all been subjects of considerable controversy here and elsewhere.

The principle of peripheral discharge, abandoned at the inception of local tube milling by reason of excessive wear on liners, went uninvestigated for many years, although the probability of its return to favor was predicted by prominent metallurgists at the time. That there was justification for this assumption has been shown recently, for as the result of trials carried out on a working scale and only lately completed at the City Deep, Ltd., a gain in crushing efficiency has been completely proved when using scoop elevators, fitted at the discharge end of the mill and passing the pulp out through the trunnion in the ordinary way.

A great deal of theoretical work has been done by various investigators in attempting to arrive at a method of comparing the crushing efficiency, or work done, by stamps and tube mills. So far, however, the production of sand of - 90 grade remains the only practical standard of comparison which has been generally adopted. As the result of exchange of ideas and experience, the points of difference in local practice have gradually been brought into line, until to-day the procedure—speaking generally—may be considered uniform for standard tube mills using the ordinary central discharge.

In feed devices, the Schmitt spiral lift is now universally adopted at the more recent plants. This appliance is particularly suitable for taking the usual free underflow from the thickening cone, and can handle without difficulty the large tonnage of broken quartz necessary to maintain the load. In tonnage of dry solids fed and percentage of moisture, practice seems to have settled down to a range of 250 to 400 tons per 24 hr., depending upon the coarseness of pulp fed, with a moisture of from 32 to 40 per cent., the latter factor varying directly as the tonnage fed, within the above limits. Local considerations, however, such as physical difference in the ore itself, the occurrence and accessibility of the gold, make it necessary to modify the procedure slightly between mine and mine. Concerning speed of rotation, as the result of tests carried out locally, the tendency has been to reduce the speed to 28 rev. per minute from the 32 to 33 considered the desideratum a few years ago, giving an

average peripheral speed, using an Osborne liner, of about 400 ft. per minute.

In regard to liners, although it has been universally recognized that these have a supremely important bearing on the crushing efficiency obtainable in tube milling, the opinions of the highest authorities the world over have been at variance as to both material and design. As far as the former is concerned, owing to the extremely abrasive nature of the ore dealt with on the Rand, practice elsewhere has had little to do with shaping the final opinion now held here, as to what is considered most suitable. Beginning with shaped silex blocks, imported with the mills, shortly afterward replaced with local chert, through various stages of composite liners, composed of cement and iron, we find the majority of tube mills on the Rand to-day using the Osborne bar liner, the standard design consisting of tapered steel bars of 4 by $1\frac{1}{4}$ in. to $\frac{3}{4}$ in. section, set longitudinally, the thick edge being held in position against the shell of the mill by wedging with flat bar iron $2\frac{1}{2}$ to 3 in. wide by $\frac{1}{2}$ to $\frac{3}{4}$ in. thick. This has proved superior to all other liners in efficiency and longevity, the extra work accomplished being greatly in excess of the increase in power consumed. Moreover, the average variation in the internal diameter of the liner during its life is much less than with either the silex or the composite-block type, which start with a thickness of 6 in. or over.

Undoubtedly the most interesting and practical development of latter-day tube milling is to be found in the results of experiments previously referred to, which had, as their original object, the investigation of the principle of peripheral discharge. The effect of obtaining such a discharge by means of a scoop or elevator is to change the nearly horizontal line of pulp level, determined by the diameters of inlet and outlet, to a sloping line from the inlet to the lowest point in the circumference of the opposite end. Thus the mill, with the same feed and pebble load, is working on a considerably smaller pulp load, with the result that a comparatively greater grinding surface is effectively exposed, resulting, as might be expected, in an increase of production of - 90 product as well as of horsepower consumed. In carrying out these trials a standard 22 ft. by 5 ft. 6 in. mill was employed, fitted with a scoop discharge, in which the radius of the lift circle could be varied. These trials were conducted in two stages: In the first, the scoop was given a maximum lift, in this case about 28 in. radius. In this series, the power consumption, wastage of pebble load, influence of feed, crushing and mechanical efficiency, were investigated. (C. E. = production of - 90 in tons per mill.

M. E. = $\frac{-90}{\text{hp.}}$.) In the second series, the radius of the lift circle was gradually reduced, and a comparison made with the previous results. In

both trials the screening used in the stamp mill was 64 to 100 meshes to the square inch. The conclusions deducible were as follows:

1. That when maintaining the pebble load at the center mark with feeds ranging from 250 to 400 tons per 24 hr., a 25 per cent. increase in crushing efficiency could be obtained, but with proportionately increased power consumption.

2. That within certain limits of feed, the weight of the pebble load can be decreased by 25 per cent. without affecting the crushing efficiency, with about a 10 per cent. decrease in power consumption, the mechanical efficiency showing a corresponding increase.

3. That the wear on the crushing load is increased 300 to 400 per cent.

4. That by decreasing the effective radius of the scoop from the maximum possible (about 29 in. in a 5 ft. 6 in. mill) to the ordinary trunnion discharge, a steady decrease in crushing efficiency is accompanied by a proportionate decrease in power consumption.

Considerable additional wear of liner would naturally result when running under these conditions; the ratio of the increase, however, was not determinable during the trials. Apart from the obvious fact that there is a considerable saving in head, it remains to be proved whether the lowering of the pulp level in the tube mill is best done by using an elevating scoop or by passing the pulp through openings in the periphery, as originally practiced by Davidson. Locally, the former system has an overwhelming advantage, owing to the serious alteration to existing plants that would be necessitated by the latter. With the use of the pulp elevator, or scoop, the enormously increased pebble feed would have to be faced, amounting to 25 tons per mill per day. This would necessitate proper additional provision being made for sorting, conveying, and feeding, which would mean a practically insurmountable difficulty in existing plants. On the other hand, the effective radius of the elevator can be reduced at will, any increase in crushing efficiency between normal and 25 per cent. being obtainable.

An interesting innovation in the design of recent plants, first introduced at the East Rand Proprietary, is the arrangement shown in Sketches C and E, Fig. 3, wherein the tube mills are placed below ground level, thus permitting flow of pulp by gravity from stamps to tube-mill cones, and saving the excessive wear on pumps. This is desirable even though it entails a subsequent higher lift of finer pulp, more suitable for pump elevation.

The latest design of tube mill used by the Rand Mines (Fig. 4) includes a ball chamber interposed between the main crushing section of the mill and the outflow trunnion. This chamber is provided with a cast-iron step lining bolted to the shell, and the grinding medium used is 10-lb. steel balls, of which about 20 are required. The object of these balls is to reduce the small spent pebbles which are being continually

rejected by the mill. The usual practice is to remove these from the pulp by means of a trommel fixed to the discharge trunnion, and periodically convey them to the stamp battery. The ball chamber is very effective in doing away with this nuisance, and has not been found to reduce the grinding efficiency of the mill, owing evidently to the additional work done by the balls, nor do the balls wear out unduly.

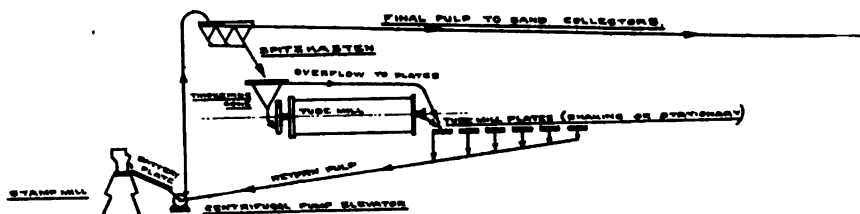
CLASSIFICATION

The necessity of providing a specially thickened and classified pulp for tube mills gave fresh impetus to the study of classification, which hitherto had been chiefly confined to the rather crude methods in vogue of separating sand from slime. The earliest classifiers were of the inverted pyramid type, a first series of spitzluten with small pockets being designed to eliminate the coarse sand and concentrate, which were collected and given a special treatment, and a second series of much larger pockets being used for separating sand from slime. A further separation of sand and slime took place in the sand collectors, the overflow of which passed to a series of return-sand spitzkasten for the further elimination of sand, the final overflow product from the latter going to the slime plant.

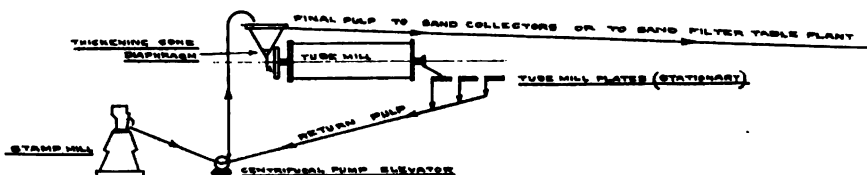
When tube mills were introduced in 1904, the coarse product was no longer separately collected but run to these mills; and the apparatus for classification gradually took the form of a simple series of spitzkasten, provided with underflow nozzles of different apertures. This underflow passed to the tube mills, while the overflow went direct to the sand collectors, in which the slime was separated from the sand and discharged by means of adjustable overflow weirs into so-called return-sand classifiers, which in turn discharged their overflow to the slime plant.

The first step in the much-needed improvement in classification was taken in 1908, when Messrs. Caldecott and Smart developed what is known as the diaphragm cone, now generally employed for thickening and classifying tube-mill pulp. This consists of a sheet-steel cone, 5 to 6 ft. in diameter and 7 to 9 ft. deep, provided near the apex with an iron disk or diaphragm. This disk is 8 to 10 in. in diameter, and the annular space between it and the sides of the cone is 2 to 2½ in.

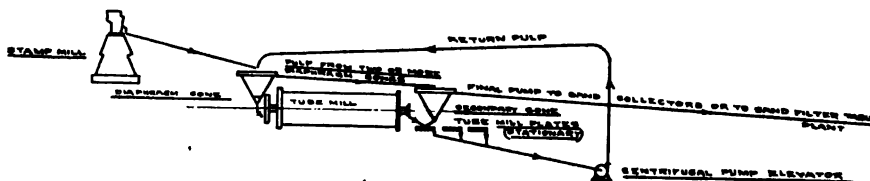
The proper action of this diaphragm is obtained only when the cone is full of solids. The mass of sand in the cone then assumes a concave surface, the deepest point from the plane of the overflow edge being immediately under the central pulp inflow. At this point, where the coarsest and heaviest product accumulates, the surface is seen to be in a state of slow continuous subsidence. This slowly subsiding mass of heavy material, conceivably irregular or conical in shape, may be presumed to act as a descending wedge, retarded in its downward course by the supporting diaphragm; while the finer sand, with the slime, tending



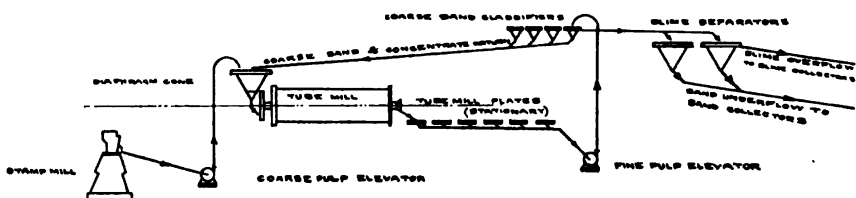
A.—Common arrangement, still retained in many plants.



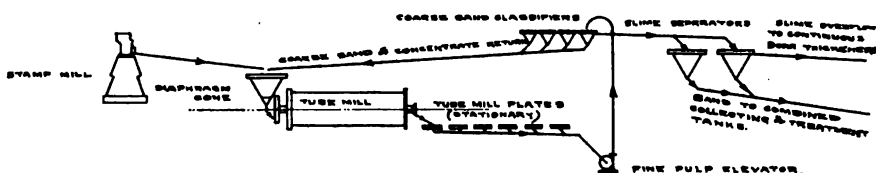
B.—Arrangement used by Consolidated Goldfields.



C.—Improved arrangement used by East Rand Proprietary and Consolidated Goldfields, the only difference being that the former use more tube-mill plates and do not use the secondary cone.

D.—Improved arrangement used by Rand Mines, Ltd.¹

NOTE—At the City & Suburban mill, battery plates are used and the pulp from the coarse sand classifiers goes to a Caldecott sand-filter plant.



E.—Improved proposed arrangement for new mill, Rand Mines, Ltd.

FIG. 3.—DIAGRAM OF VARIATIONS IN CLASSIFICATION ARRANGEMENTS.

to adhere to the sides of the cone, would appear to be literally pushed aside and crowded upward by the central column, so reaching the overflow rim and escaping. The slow, thick stream issuing from the apex carries only from 26 to 30 per cent. moisture, and a minimum of the fines which it is desirable to exclude from the tube mill.

These cones have a large capacity, and, barring the one disadvantage of vertical height required, may be considered the most simple and suitable means yet devised for providing all the conditions requisite for a tube-mill feed that will enable the mill to work at highest efficiency. This device is now used either as the sole means of classifying mill pulp, as in B, Fig. 3, or in conjunction with hydraulic classifying cones, as in D and E, Fig. 3. An important aspect of this innovation was its having made possible the introduction, in 1907, of the Caldecott sand-filter table, a device whose success obviously depended upon securing a suitable thickened pulp, containing a small amount of water and a minimum of slime. The primary object of this appliance, as explained by the inventor,³ was to obtain sand in such a condition for treatment as would warrant the elimination of sand-collecting vats, which could then be used for treatment purposes. Obviously, the effect of this was to increase very considerably the capacity of a leaching plant.

Metallurgically considered, however, the significant feature of this appliance was its revival of the old question of the possibility of single treatment of sand after proper classification. In this connection it is interesting to note a prediction made by Charles Butters in 1895, that "the whole question of double treatment really resolved itself into a matter of filling the vats with clean stuff, and he was confident that the day would come *when there would be no double treatment.*" The first notable success in America in collecting and treating sand in the same vat was at the Homestake mill; but in Africa, with the single exception of the East Rand Proprietary, double treatment has been retained until quite recently.

In 1910, when the new plant for the Modderfontein B. mine was being designed, I undertook to evolve a simpler method than the filter table for obtaining a clean sand, with a view to the subsequent elimination of separate collecting tanks. This classifying plant consists of eight small primary hydraulic cones, 2 ft. 9 in. in diameter and 2 ft. 6 in. deep (see D, Fig. 3), designed as concentrators for insufficiently ground sand particles from the tube mills. The overflow from these gravitates to four larger hydraulic cones, 8 ft. in diameter by 6 ft. 9 in. deep, which effect a very satisfactory separation of sand from slime, the overflow gravitating direct to the slime collectors. The underflow of the large cones is evenly distributed in the collectors by the Butters and Mein

³ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. x, No. 2, p. 43 (Aug., 1909).

distributor, a device recently revived on the Rand Mines group after several years of disuse.

This system, in view of the possibility of treating a considerable amount of fine (-200) sand in the Butters filter, was found to be well

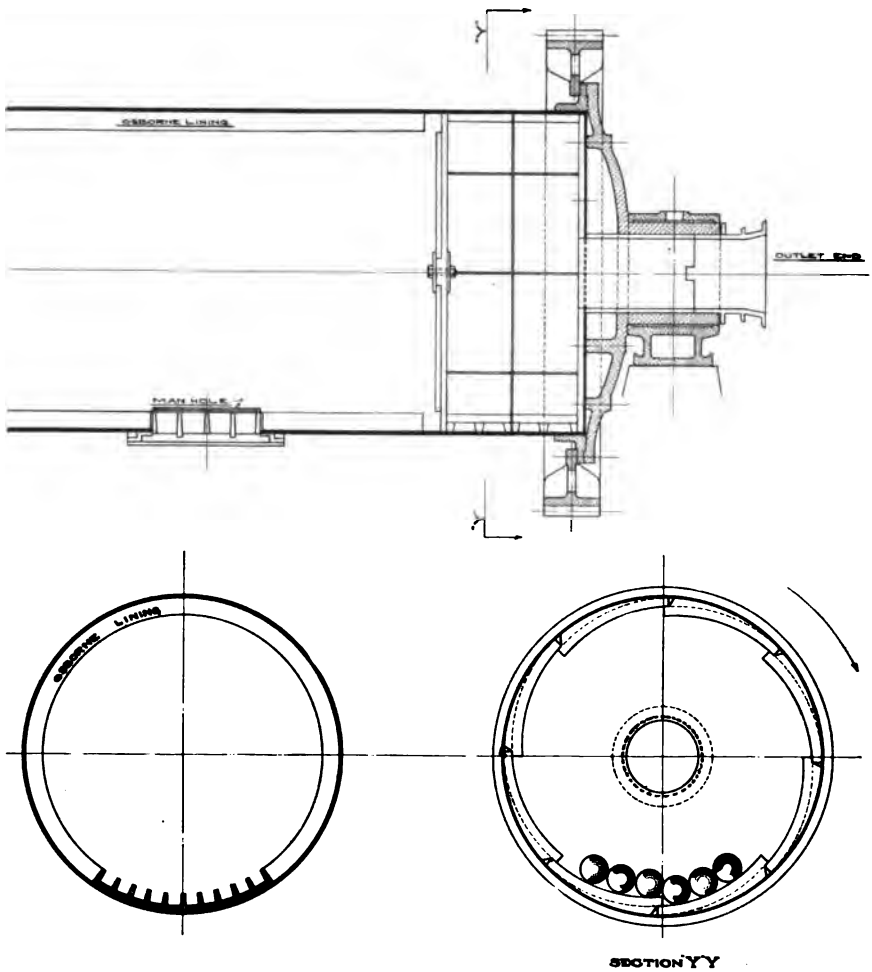
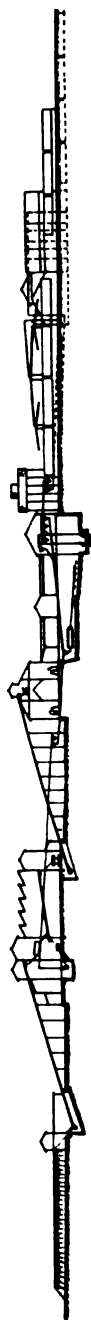
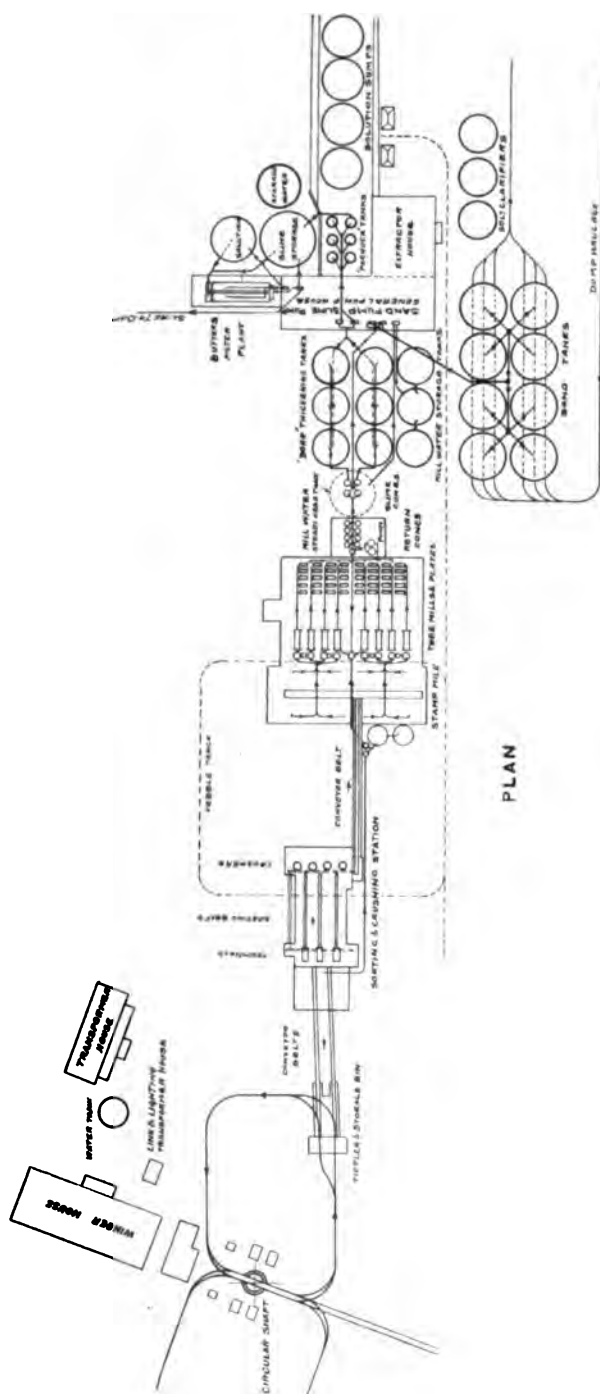


FIG. 4.—TUBE MILL WITH BALL CHAMBER, MODDERFONTEIN B. REDUCTION WORKS, RAND MINES, LTD.

adapted to this mode of filtration, the correct proportion of fine sand and slime being easily obtainable. Moreover, an evenly distributed sand charge was secured, free from lumps and layers of slime. This system has been adopted by other mines of the Rand Mines group; and in newer plants has, with a few modifications, superseded the primitive method of



ELEVATION



PLAN

FIG. 6.—PLAN AND ELEVATION, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

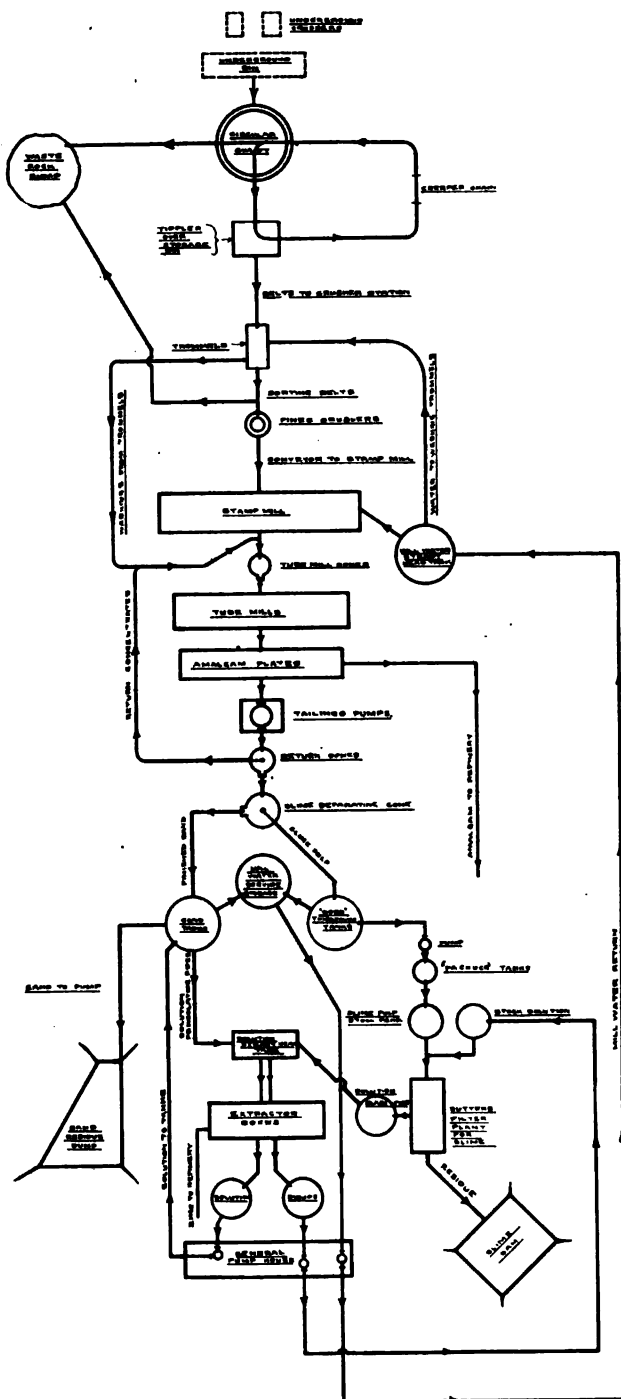


FIG. 7.—FLOW SHEET, PROPOSED NEW REDUCTION WORKS, RAND MINES, LTD.

charging sand and slime together into collectors and depending upon a Kaffir and a movable hose for even distribution.

At the Crown Mines, an improved hydraulic attachment for cone classifiers was devised by H. Brazier, the reduction foreman at "C" mill. This consists of an adjustable nozzle for discharging water in proximity to the apex of the classifying cone in the form of a thin circular sheet, directed horizontally between the nozzle and the cone. By means of this system of cone separators, the classification of sand is permissible within a wide range, and has enabled us to obtain a charge of sand so permeable and uniform that after practical trials the separate sand collector is now recognized to be superfluous, and will be omitted in forthcoming plants to be erected by the Rand Mines group (Figs. 5, 6, and 7).

We have then at the present time on the Rand two very satisfactory methods for classifying sand: (1) the Caldecott sand-filter table, and (2) the system of hydraulic cones just described. The former possesses the advantage of delivering a solution-borne sand to the treatment tank, and so shortening the time of treatment and increasing the capacity of existing plants, which is a very desirable thing, especially where the saving of capital outlay is imperative, as at the City & Suburban, where the sand-filter table obviated an expensive and awkward extension of plant. But for new mills adopting single treatment of sand, the indirect saving in sand plant would appear to be offset by the lower initial expense of the hydraulic-cone system itself, the small amount of attention required, and its negligible cost of maintenance and operation, as compared with the filter table.

In E, Fig. 3, an improved arrangement of the two series of cones is shown. Differing from general practice, the slime separators are placed near the ground, only just high enough to permit their overflow to run to the Dorr thickeners, while the underflow of sand is elevated to the sand-treatment tanks. We thus have but two lifts in this type of plant: the first, and highest, elevating the tube-milled pulp to the primary classifiers; and a much shorter lift elevating sand and water only. This may be seen in Fig. 7, where, however, the treatment tanks are diagrammatically shown at a considerable elevation, whereas they will stand only just high enough above the ground to allow room for the residue trucks.

TREATMENT BY CYANIDE

It was only after the introduction of the cyanide process that the distinctive terms "sand" and "slime" came into common use. In the early days of cyaniding the bugbear was slime, or the unleachable, finely divided portion of the mill pulp, which the mill man endeavored to avoid making, in order to secure as large a proportion as possible of leachable sand. Until a method of treating slime was devised, this

product was impounded in dams; these accumulations have now mostly been treated.

When the decantation process made it possible to deal with current slime direct from the stamp mill, the production of the latter was no longer avoided. Until vacuum filtration was introduced, the slime represented between 30 and 40 per cent. of the total product of crushing; but this proportion has gradually increased with the progressive improvements in treatment, until now as high as 60 per cent. of the pulp is treated as slime. We may say, therefore, that tube mills, improved classification, and the vacuum filter have made possible (1) the treatment of a greater proportion of slime than formerly, which has improved the total extraction, and reduced the cost of treatment, since slime can be more cheaply treated than sand, and (2) the treatment of more finely crushed sand, with consequent improvement in extraction. In the best plants, a recovery of 90 per cent. of the gold from sand and 93 per cent. from slime is now being obtained, or a total recovery of 96 per cent., which is probably the highest extraction economically attainable on the Rand. The metallurgist in Johannesburg to-day is therefore chiefly concerned with those improvements in appliances, general technique, and administration, which, in view of the much reduced grade of ore, will minimize cost of treatment.

TREATMENT OF SAND

The treatment of the sand by leaching with cyanide solution in steel tanks, ranging in size between 45 and 56 ft. in diameter, is still the accepted practice. In the early days an attempt was made to collect and treat sand in the same tank, but an imperfect knowledge of classification, or a failure to recognize its importance, made this scheme impracticable. Until very recently, the prevailing practice was to collect the sand in a series of tanks, known as "collectors," from which it was shoveled out by natives and transferred to the treatment tanks. The majority of plants were built with the collector superimposed on the treatment tank, but this arrangement was finally superseded by the erection of collectors and treatment tanks on the same level, the transference of sand from one to the other being by means, first, of trucks, and later, by belt conveyor. The latter was supposed to be cheaper in first cost, as it eliminated the expensive steel superstructure for supporting the enormous weight of superimposed tanks, and also reduced the height of pulp elevation. But it would appear now, after a pretty thorough experience with both systems, that, as regards first cost, the superimposed system has a slight advantage; while owing to the rapid disintegration of conveyor belts in the dry atmosphere of the Rand, and the considerable maintenance cost of conveyor systems in general, the superimposed

tanks are less expensive to operate, in spite of the higher pulp lift required.

At the present time there is still considerable variation in the modes of collecting sand. The following methods are in use:

1. In the older plants all the mill pulp is run into a collector, through a rubber hose 4 to 6 in. in diameter, manipulated by a Kaffir who moves about in the tank changing the position of discharge to prevent undue slime accumulations; the slime and water, with varying proportions of fine sand, overflow through discharge gates provided with an adjustable canvas blind, which is raised to suit the overflow as the tank fills.

2. The total mill pulp is distributed to the collector by means of a peripheral launder provided with outlets; the water and slime overflow through an adjustable opening at the center.

3. The sand is classified and thickened in diaphragm cones, dewatered on a sand-filter table, and delivered in cyanide solution either to a collecting or treatment tank by means of a Butters and Mein distributor.

4. The sand is classified in hydraulic cones and delivered in water to a collecting tank by means of a Butters and Mein distributor; in the plant shown in Fig. 7, the sand will go direct to the treatment tank.

It was formerly customary in some plants to give the sand a preliminary treatment with a weak cyanide solution in the collectors; this practice has now been generally abandoned, and the only operation that takes place in the collector is the forced drainage of the charge by means of pumps, thus reducing the moisture to about 14 per cent. The sand is discharged from the collectors by (1) hand shoveling through discharge doors into treatment tanks or to belt conveyors; (2) by means of the Blaisdell excavator. The high capital outlay required for the latter, without compensating economy in operation, has led to the retention in newer plants of the older system of hand shoveling by natives. All plants using a belt conveyor from collectors to treatment tanks have, however, retained the excellent Blaisdell distributing mechanism for distributing the sand in the treatment tank.

The system of applying cyanide solution to the sand does not differ essentially from practice elsewhere, and need not be particularized here. The standard strength of strong solution used ranges between 0.10 and 0.25 per cent., depending upon local conditions, the tendency in recent years being to use weaker solutions than formerly. From 6 to 8 days' contact is usually allowed in the treatment tanks, and about 2 parts of solution to 1 of ore is the average quantity required for leaching purposes.

The methods in vogue of disposing of the residue are by hand shoveling or excavation with the Blaisdell machine into (1) trucks, (2) Bleichert aerial conveyor, or (3) belt conveyor. The second method is used at

only one property, the Brakpan mines, where it is reported to be giving satisfaction; the third method is used on two mines, but whatever advantages it may possess have not been generally recognized; while the first method, namely, truck haulage by surface cable, is in general use. It is considered the cheapest, most flexible, and best adapted to local conditions, and will probably continue to be the favorite means of disposing of sand residue.

TREATMENT OF SLIME

Since 1894, the continuous treatment of slime by the so-called decantation method has meant the recovery of over 80 per cent. of the gold from a very considerable portion of the total ore crushed, and has been retained by the majority of operating mines. In most cases, however, it has been retained, not because its limitations have not been recognized, but because possible improvements in recovery do not always justify the sacrifice of capital involved in replacing old plants with new, especially on mines of short remaining life.

The technique of decantation, which is well described in several technical works, need only be briefly touched upon here. The slime pulp from the classifiers is settled in what are known as collectors, large cone-bottom tanks, provided with an overflow rim and adjustable decanter, in which the slime settles to the bottom, the water escaping at the overflow rim. The settled slime charge after decantation, containing approximately 50 per cent. of moisture, is then sluiced out with a weak cyanide solution into the intake of a centrifugal pump connected with the center of the cone bottom, and transferred to another tank, known as the "first settlement tank." Here the charge is kept in circulation by means of a pump, then allowed to settle, as in the collector, and the gold-bearing solution is decanted off to the zinc boxes. This decanted charge is again transferred, subjected to a similar settling and decantation in a "second settlement vat," and finally discharged to the dam with water or a very dilute precipitated solution. Settlement in these tanks is hastened by the use of lime, which is applied in a variety of ways, being either periodically fed to the stamps or tube mills, or to the slime pulp in slaked form, by means of feed hoppers, or as milk of lime from a grinding pan.

The limitations of decantation are obvious: The displacement of the dissolved gold is only practicable within certain economic limits, so that a certain definite gold loss in the final discharge to the dam must always be reckoned on. This loss is necessarily variable, depending upon capacity, difficulties in settlement, and general design of plant. On the other hand, the process is simple and comparatively inexpensive in operation, and, in spite of the losses through imperfect washing, is peculiarly applicable to the low-grade slime of the Rand. Even the advocates

of more exact and positive methods of treatment must admit that in the latest and best designed decantation plants, in which large capacity is allowed for settlement and dilution, and all other conditions are favorable, the margin between results so obtained and the net results of filtration is not a wide one.

The introduction of the so-called Usher process was an attempt to improve upon decantation by removing the gold-bearing solution by means of continuous upward displacement. The treatment tanks were provided near the bottom with a large number of perforated pipes radiating from a central upright pipe. The precipitated solution was introduced into this central main from a steady-head tank and allowed to issue slowly through the holes in the radial pipes into the bottom of a partly settled charge. The speed of flow was just sufficient to keep the slime in suspension and permit of a clean overflow at the tank rim. The chief advantage claimed for this system was the reduction in plant required by reason of the time saved in avoiding decantation and settlement. But certain insuperable mechanical difficulties developed in connection with this process which offset its alleged advantages, and it was subsequently replaced by either decantation or filtration.

The first important innovation in slime treatment was the introduction of the Butters vacuum filter in 1910. The essential features of this device are too well known to require description here. There was from the first no question as to the additional recovery obtainable by the filter; the doubtful point was whether the cost of operation would offset the gain in recovery to the extent of nullifying its advantages. This, however, has not proved to be the case. The prevailing cost of operating the filter on the Rand is from $2\frac{1}{2}$ d. (5c.) to 4d. (8c.) per ton of slime filtered, depending upon tonnage treated; and the additional gold recovered ranges from 6d. (12c.) to 2s. (48c.) per ton, depending upon the efficiency of the decantation plants superseded by the filter.

The vacuum filter is now generally admitted to be applicable in the following cases: (1) Where it is desirable to extend a decantation plant of inadequate capacity or obsolete design; (2) where difficulties in settlement make it impracticable to use decantation without prohibitive extension of plant; (3) in all new plants. The most notable instances of (2) were the case of the Randfontein Central mill, where a comparatively new decantation plant treating 2,000 tons of slime per day was replaced by a Butters filter, resulting in a considerably increased recovery of dissolved gold; and the case of the Robinson Gold Mining Co., where the expenditure of £32,000 in 1911, on filter and Pachuca agitators, resulted in an increased net profit of £2,500 per month.

The Butters filter has been adopted in all new plants erected within the last five years. The following Rand mines have adopted the filter:

Butters Filter Plants in the Witwatersrand District

Plant	Number of Filter Leaves	Capacity, Tons per Day
Crown Reef.....	310	900
Brakpan.....	348	1,000
New Modderfontein.....	348	1,000
Bantjes.....	240	600
Randfontein South.....	89	250
Robinson.....	468	1,350
Modderfontein B.....	135	550
Langlaagte Estate.....	117	350
Crown Reef Extension.....	125	400
Randfontein Central.....	532	2,000
Van Ryn Deep.....	310	1,000
Geduld.....	190	500
Government Areas.....	412	1,300
Modderfontein Deep.....	250	700

The slime treatment in these modern plants consists in settling the slime in the same type of settlers as is used in decantation, namely, in tanks as large as 56 ft. in diameter, provided with cone bottoms, decanters, and overflow launders. When the charge is settled and decanted down to about 50 per cent. moisture, it is transferred with cyanide solution to the Pachuca tanks. These tanks are of standard size, 15 ft. in diameter and 45 ft. deep, and hold between 80 and 100 tons of slime (dry weight), depending upon dilution. This agitator, requiring air representing in volume and pressure an expenditure of about 3 hp., is considered, by reason of its simplicity and low operating and maintenance cost, to possess advantages over the various mechanical types, in spite of its great height. Agitation is continued from 6 to 8 hr., and the pulp is then transferred to a large storage reservoir, whence it is delivered to the Butters filter plant, as required.

Two modifications of this procedure will be adopted in the next plant to be erected by the Rand Mines, Ltd. The old system of intermittent slime settlement will be replaced by continuous settlement in Dorr thickeners. In trials conducted with this device it was found impossible to reduce the slime to the same low moisture as the intermittent system, without a much larger capacity than is allowed in American mills, where this device is chiefly applied to the much more simple settlement of slimed mill pulp containing a considerable quantity of fine sand. To compete with intermittent slime settlement on the Rand, a Dorr thickener 40 ft. in diameter cannot handle more than 150 tons of slime per day; but even so, in view of the labor and power consumption involved in handling intermittently settled charges, and the high capital cost of slime settlers, the continuous system would appear to possess distinct advantages.

The continuous agitation of slime is made possible by the system just described; and is particularly well adapted to the newer plants where the

height of the Brown agitators can be utilized for the required gravity flow into the Butters stock tank. This system of allowing pulp to flow slowly through a series of agitators was first applied in Mexico; its first application on the Rand was at the East Rand Proprietary about two years ago. It seems likely that all plants to be erected in the future will adopt continuous slime settlement and agitation, followed by vacuum filtration.

PRECIPITATION

The use of filiform zinc for precipitating gold was introduced on the Rand by MacArthur in 1890. At first no difficulty was experienced in the deposition of gold from the stronger cyanide solutions required for sand treatment; but when it was found possible to recover gold from slime with much weaker solutions, precipitation on zinc became more difficult. It was at this time that the Siemens-Halske electrolytic process, which was more effective than zinc in dealing with these weak solutions, threatened to replace the older method; but owing to serious defects in operation, already touched upon, it was finally abandoned in favor of the zinc method, which had been rendered much more efficient by the immersion of the shavings in a solution of acetate of lead, preparatory to filling the extractor boxes. With this exception, precipitation practice on the Rand does not differ materially from that in other mining districts where filiform zinc is used.

The zinc shavings are cut in the usual manner to a thickness of about $\frac{1}{500}$ in. and 1 cu. ft. of such filaments loosely packed, per ton of solution per 24 hr., is the average allowance for capacity of extractor boxes. These boxes, usually built of steel, are from 4 to 6 ft. wide, and of corresponding depth, with 6 to 10 compartments. The existing system of gold precipitation, though highly efficient on the dilute solutions used on the Rand, admittedly possesses many awkward features which have for years stimulated investigation with a view to devising a more compact, positive, and less wasteful substitute. Its weak points are (1) the great area required for plant; (2) the labor required in dressing and cleaning up extractor boxes; (3) the uncertainty of cleanup, owing to the variable distribution of gold not immediately recoverable; (4) the impossibility of recovering at once all gold deposited within a given period; (5) the enormous loss of zinc in the destructive process of recovering its gold content.

The Merrill zinc-dust method, as perfected in America, offered certain distinct advantages over the older process. It is neater and more exact in operation, requires less labor, and possesses the very attractive feature of yielding a complete cleanup of all the gold deposited. After practical trials this process was introduced at three Rand mines: the Brakpan, New Modderfontein, and Modderfontein B.

At New Modderfontein the conditions of precipitation are somewhat peculiar, and imposed a very severe test on the Merrill process. At this plant a solution of under 0.01 per cent. in KCN is effective in dissolving gold in the slime. The resultant gold-bearing weak or slime solution is comparatively simple in composition with a very low metallic and cyanide content, which rendered it extremely difficult to precipitate satisfactorily on zinc dust without undue additions of cyanide and lead acetate. Various measures were tried to promote galvanic action and enliven this inert solution, but without success; and at this plant it was finally decided to revert to the older zinc-shaving method. The process has, however, been retained at the other two properties mentioned.

At Modderfontein B., only one of the two 40-frame, 52-in. Merrill presses installed is required to precipitate 1,900 tons of solution per day—an unusually high duty. The average gold content of the weaker solution treated is 2 dwt., and its cyanide strength 0.018 per cent. The consumption of zinc for the last six months at this mine (June to November, 1914, inclusive) was 0.180 lb. per ton of ore milled, or 0.145 lb. per ton of solution precipitated.

At Brakpan, three 40-frame Merrill presses are in use, precipitating about 2,800 tons of solution per day. Here the consumption of zinc during the last three months (September to November, 1914) is given as 0.152 lb. per ton of solution precipitated and 0.251 lb. per ton of ore milled. The following details covering these three months' operations have been kindly submitted by the Brakpan Mines, Ltd.:

	Tons Precipitated (Average per Day)	Gold Content of Inflow	Gold Content of Outflow	KCN Content Inflow	KCN Content Outflow	Alkalinity (CaO) Inflow	Alkalinity (CaO) Outflow
		Dwt.	Dwt.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Sand solution.....	269	3.952	0.045	0.070	0.036	0.023	0.027
Medium sand solution.	769	1.548	0.056	0.017	0.009	0.0146	0.015
Slime solution.....	1,791	1.271	0.048	0.010	0.007	0.0149	0.015

The consumption of zinc in the Merrill presses is approximately $\frac{1}{10}$ lb. less per ton milled than in extractor boxes. The cost at Johannesburg of cut shavings is 4.2d. per pound, of zinc dust 3.93d. per pound, so that as regards zinc consumption the presses have the advantage. But this is offset by the cost of additional cyanide required to strengthen the slime solutions sufficiently for good zinc-dust precipitation, which was found to be, at New Modderfontein, 1d. per ton milled.

The general opinion in regard to zinc dust, after three years' experience with the process, is that in effecting economies in zinc consumption and labor, and in affording a complete cleanup of gold, it has fulfilled the claims made for it. On the other hand, it requires more vigilance and

care in manipulation than the zinc-shaving method, is liable to erratic fluctuations in efficiency without assignable cause, and requires the use of stronger solutions than are actually needed for dissolving purposes.

One cannot escape the conclusion that zinc in any form is far from being the ideal precipitant for gold. In 1913, about 9,000,000 lb. of zinc were consumed by the mines of the Rand. When we consider that the greater part of this irrecoverable loss is due to the destructive method employed in separating the gold from the zinc after deposition, it is evident that the existing system is an extremely wasteful one. For this reason, the subject of gold deposition presents one of the most profitable fields for investigation in the whole realm of metallurgy, and I venture to predict that in this stage in the reduction of gold ores, the most important advances in the future will be made.

CLEANING UP AND SMELTING

Local methods of cleaning up the zinc-gold precipitate do not differ essentially from practice elsewhere. The difference in practice between local mining groups is one of minor details only, depending mainly on design and capacity of plant. Briefly, the procedure consists in periodic removal of zinc-gold "slime" and broken zinc or "shorts," and the renewal of the displaced zinc by fresh shavings. The amount of zinc removed from the boxes at each cleanup depends upon the balance desirable between monthly realization of gold and zinc consumption. A minimum of gold reserve in extractor boxes is attainable only at the expense of zinc consumption; hence the recorded fluctuations in the latter between 0.30 lb. and 0.50 lb. per ton of ore treated.

The usual method of destroying the zinc in lead-lined tanks with sulphuric acid is in use on the Rand. A cheap source of sulphuric acid now in common use is the bisulphate of soda (NaHSO_4), a by-product from the local manufacture of sulphuric acid used in making nitroglycerin. It contains about 30 per cent. of available sulphuric acid. Filter presses are in common use for removing the zinc sulphate from the gold slimes, and for the subsequent washing with hot water, preparatory to drying and smelting.

The two methods of smelting followed on the Rand are (1) direct crucible smelting and (2) lead smelting. The former possesses the important advantage of eliminating the gold absorption which takes place in the furnace bottom of the Taverner or lead furnace, and which is not recoverable until it is necessary to rebuild the furnace; on the other hand, the lead furnace costs somewhat less to operate. Both systems have their good points, and both will probably continue to be used as long as zinc is retained as the precipitant for gold.

PERCENTAGE EXTRACTION AND TREATMENT COST

In concluding this paper a few remarks on extraction and cost of treatment on the Rand may be of interest. It is significant of the steady, consistent progress made in metallurgy during the last two decades, that the recovery of gold has risen from 60 per cent. or thereabouts (1890) to a maximum of 96 per cent. (1914). The chief contributing factors were: The introduction of slime treatment (1894); the advent of the tube mill (1904); improvements in classification (1907 to 1910); and finally the introduction of the vacuum slime filter (1909). Concurrently, many improvements in detail, both mechanical and metallurgical, have had much to do with the general advance.

Similarly, in regard to the steady reduction in cost of treatment, many influences have been at work, such as improved efficiency achieved in crushing with heavy stamps and tube mills, improvements in design of plant and change in supervision and general organization. The cost of reducing ores on the Rand ranges between 2s. 10d. (68c.) per ton milled and 5s. (\$1.20), depending upon capacity and efficiency of plant. As an instance of low reduction cost I have chosen the case of Modderfontein B., a modern, medium-size plant, showing first the average cost during 1913, when the tonnage was comparatively small, and during November, 1914, when the capacity of the plant had been increased.

Cost of Ore Treatment at Modderfontein B.—Average for 1913

Average Tons per Month Milled = 33,715

	Per Ton Milled Pence	Cents
Ore sorting and breaking:		
Including white wages, colored wages, colored labor sundries, machinery and spares, sundry stores, workshops, power, and sundries	3.429	6.8
Transport from crusher station to mill bins:		
Including white wages, colored wages, colored labor sundries, stores, workshops, power, and sundries.....	0.918	1.8
Stamp milling:		
Including white wages, colored wages, colored labor sundries, mill spares, shoes and dies, water, sundry stores, workshops, power, and sundries.....	8.118	16.2
Tube milling:		
Including white wages, colored wages, colored labor sundries, liners, tube-mill spares, water, sundry stores, workshops, power, and sundries.....	7.890	15.8
Amalgamation:		
Including white wages, mercury, sundry stores, workshops, assaying, retorting, and smelting.....	2.163	4.3
Cyaniding:		
Including white wages, colored wages, colored labor sundries, water, cyanide, lime, zinc, chemicals, trucks, rails and fittings, conveyor belts, sundry stores, tailing contractor, workshops, power, assaying and smelting, stables, and sundries.....	19.554	39.1
Total.....	3s. 6.07d.	84.0c.

Cost of Treatment at Modderfontein B.—November, 1914

42,000 Tons Milled

	Per Ton Milled Pence	Cents
Ore sorting and breaking.....	2.646	5.3
Transport from crusher station to mill bins.....	0.349	0.7
Stamp milling.....	8.754	17.5
Tube milling.....	6.251	12.5
Amalgamation.....	1.834	3.6
Cyaniding.....	14.531	29.0
Total.....	2s. 10.4d.	68.6c.

In closing this necessarily limited survey of Rand metallurgical practice, I must acknowledge having referred freely to those excellent sources of technical information, the *Transactions of the Chemical, Metallurgical and Mining Society of South Africa*, and the very complete *Text Book of Rand Metallurgical Practice*. I wish also to express my obligation to K. L. Graham and J. R. Thurlow of the Rand Mines, Ltd., Metallurgical Staff, and to the Mechanical Engineering Department, for assistance rendered in the preparation of this paper.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Cost Factors in Coal Production

BY WILLIAM H. GRADY, BLUEFIELD, W. VA.

(New York Meeting, February, 1915)

FACTORS entering into the market value of coal are its grade, and the cost of labor, material, and capital. Reduction in these costs cannot be expected in the future, and it therefore follows that greater economy in their use must be accomplished if it is desired to hold the present markets; more tons of product per unit must be obtained in a given time from both men and material, and means devised for increasing the percentage of lump and domestic sizes.

This paper is presented for the purpose of inviting discussion and stimulating thought in regard to the influence of methods of procedure and plans of mining, upon the quality of coal, the economical use of labor, material, and capital, and the cost of production. Even the better methods of the present day leave much to be desired, and a review of the reasons and necessities for their use may be of value in pointing the way to more economical mining. The writer does not attempt to discuss untried methods of mining, but rather to state clearly and concisely a summary of the results obtained under the several methods of procedure that have been adopted.

In some instances a high degree of concentration has been effected, and comparisons of the results obtained with the results of lesser degrees of concentration form an interesting study. It is believed that much may be accomplished in improving the quality, and in the more economical use of labor, material, and capital, by *concentration*, and it is to concentration in particular that your study is invited.

The writer, who has occasion to visit many mines, most of which are under different management, has been inclined to the opinion that improvement has been retarded in many ways and that mining has not as yet been entirely freed from the early-day practices which were forced upon it.

Practices resulting in lack of concentration, to which reference will be made, are: The absence of robbing, necessitating the frequent interposition of barrier pillars; lack of system and of proper supervision, resulting in losses of life and coal. These practices could not well be

avoided years ago, for with coal on the railroad cars selling at practically the cost of production, the incentive was to mine only the coal which could be produced at a profit, regardless of what the future cost might be.

The writer, in talking with some pioneer operators in West Virginia, was much impressed to learn that in the early '80s, not only were pillars not robbed in some mines, but many of the operators firmly believed that the extraction of the pillars was a physical impossibility. Quite as much impressed were these operators when it was stated to them that to-day, in some mines, over 95 per cent. of the coal in the seam was being extracted, and that pillar coal, if properly mined, is of the same quality as room coal, and cheaper, both to the operator and the miner, to produce.

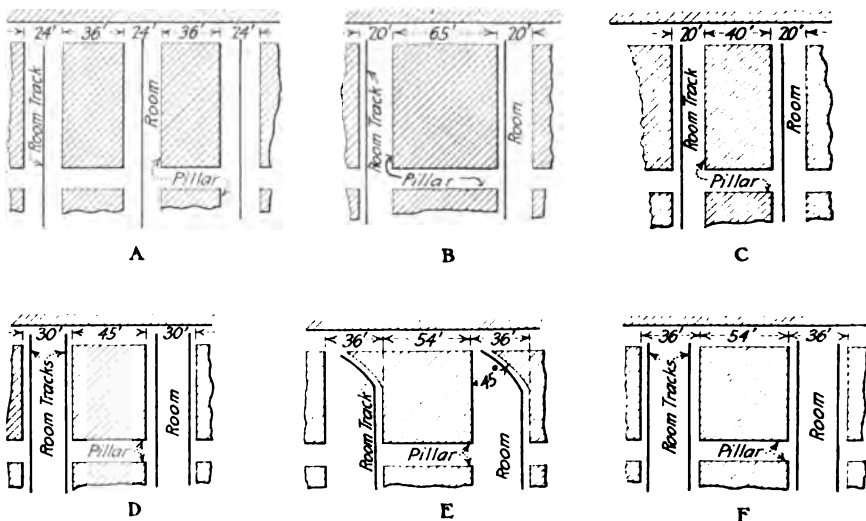


FIG. 1.—TYPICAL METHODS OF PROCEDURE IN COAL MINING NOW IN EFFECT, UNDER CONDITIONS WHICH ADMIT OF COMPARISON.

It may be stated that the unit with which we have to deal is the room; what takes place at its face is the real productive work of the mine, and all else underground is for the purpose of serving best the worker at the room face. Fig. 1 shows several typical methods of procedure that have come under the writer's observation. They are of particular interest in that one may see them in mines following the same plan, working the same seam, under conditions which admit of comparison. The features of these methods are given in Table I.

In all of these methods variations may be seen, from entries driving with no rooms turned to entries driving with two or more rooms turned and driving as the entries advance; in respect to the robbing, one may see variations from robbing following immediately upon the comple-

tion of the first two rooms to the robbing following at an indefinite date after the completion of the first workings of the panel. Where continuous paneling, or advancing robbing, is in effect, robbing is not compelled to wait until the completion of all the entries of the panel.

TABLE I.—*Methods of Procedure*

Sketches	A	B	C	D	E	F
Width of room in feet.....	24	20	20	30	36	36
Width of pillar in feet.....	36	65	40	45	54	54
Location of track.....	In center of room	Along robbing rib	Along robbing rib	Along robbing rib	Along robbing rib	Along robbing rib
Location of gob	Along both ribs	Opposite robbing rib	Opposite robbing rib	Between tracks	Opposite robbing rib	Between tracks
Number of men per room....	1 to 2 rooms	1	1 or 2	2	6	4
Feet of room face per man	48	20	20 or 10	15	8.5	9
Feet of entry per man....	120	85	60 or 30	37.5	15	22.5

The number of rooms per entry varies from about 12 to an indefinite number, and the depth of the room varies from about 300 to about 800 ft. The amount of timber and the manner and time of placing same depend largely upon the individual miner, and as a rule there are no specific instructions for his guidance; also, in general, no effort is made to recover the timber in robbing.

A method of procedure observed by the writer (but which has not as yet been sufficiently tested out in the matter of recovering the pillar to warrant its unreserved adoption), is shown in Fig. 1, *E*. Here it is intended that rooms shall be driven 36 ft. wide on centers 90 ft. apart, carrying a room face at an angle of 45° and a single track along the robbing rib but curved to parallel and follow the length of the room face. It is intended to work six men to the room, the gathering motor receiving and placing three cars at a time. Immediately upon the completion of the room the pillar is to be withdrawn. By this method of procedure a high degree of concentration will be effected and the efficiency of the gathering motors, mining machines, and miners will be increased. It is also hoped that by carrying the working face on a diagonal, fewer unexpected falls of top will occur than at present, because the fracture will generally be partly exposed before the entire coal support is removed from beneath it.

The relative degree of concentration effected in the above methods of procedure is shown in Fig. 2. The units of measure adopted for comparison are linear feet of room face per man and linear feet of entry per man.

In most mines the miner at the face is responsible for the safe working conditions of his room. In the above methods of procedure, therefore one might say that, for the same expenditure of time, energy, and watchfulness, the relative degree of security which the miner may feel as a result of his efforts is inversely proportional to the room space he occupies. It is also true that for cars of the same height the energy expended by the miner, or the work done in loading the coal, is much less where two or more men work per room and the room space per miner is low, than where one man works per room and the room space per miner is high.

In the grouping of rooms as outlined in Fig. 1 many arrangements were made from which have matured certain well-defined plans. Prob-

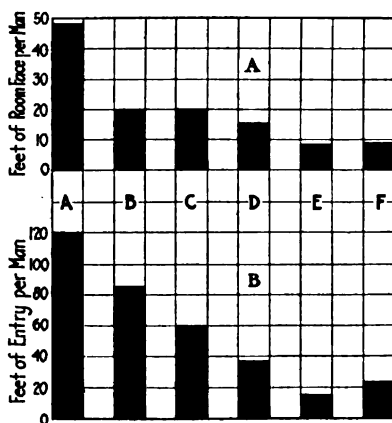


FIG. 2.—GRAPHICAL COMPARISON OF THE RELATIVE DEGREE OF CONCENTRATION IN THE METHODS OF PROCEDURE SHOWN IN FIG. 1.

ably the consensus of opinion favors the panel system, but even with it there are differences of opinion. Many men think that the entries should be driven to the inside lines of the property and the coal extracted retreating; others think that half of the property should be worked advancing and the remainder retreating; yet others think that all or nearly all of the coal should be extracted as the entries advance. It is the writer's opinion that the coal should be extracted in such a manner that the present worth on the returns from the mining venture will be greatest, both to the property owner and the operator, leaving only such coal during the advance of the entries as will permit of profitable mining until the final exhaustion of the property. Figs. 3 and 4 show typical plans on the panel system. Fig. 3 is the square or rectangular panel, Fig. 4 the continuous panel.

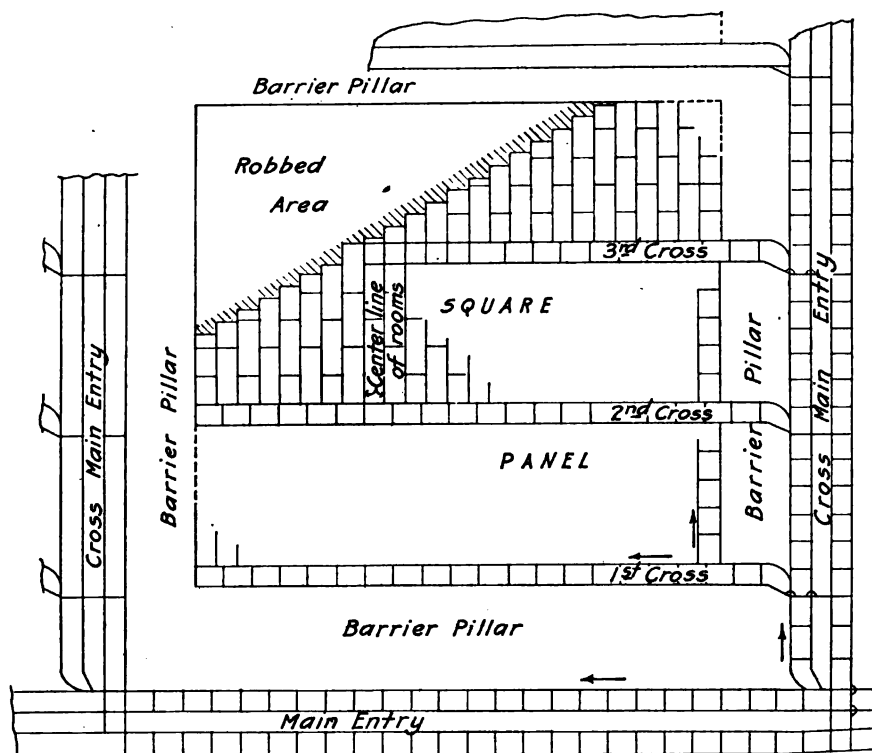


FIG. 3.—TYPICAL PLAN OF MINING ON THE PANEL SYSTEM.

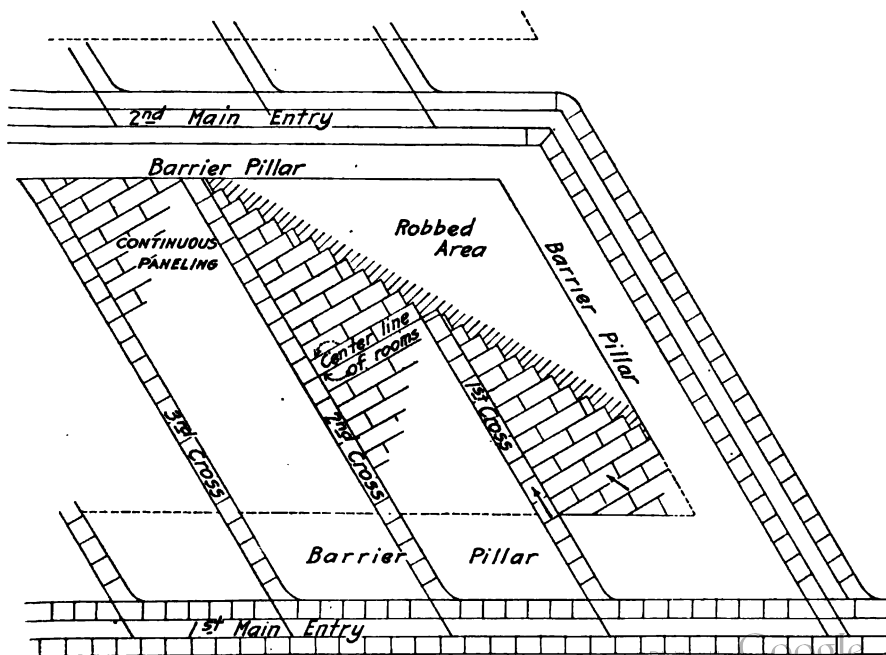


FIG. 4.—TYPICAL PLAN OF MINING ON THE PANEL SYSTEM.

Fig. 5 represents a typical lease or property of 1,000 acres from which it is desired to produce about 2,800 tons per day when running at a maximum. It is assumed for the purpose of this discussion that the coal is fairly clean, 6 ft. thick, and the condition of grades, top, and bottom fair. It is also assumed that the rate of loading per man per day is 16 tons. The questions to be decided are: What method of procedure and what plan are best? In order intelligently to make a decision the following information is desired:

1. What period of time will be required to reach the output?
2. How many day laborers, mining machines, mine cars, mules for gathering, and main-haulage motors will be required?
3. How much main entry, main entry track and trolley wire; cross main entry, cross main entry track and trolley wire; room entry, room-entry track, and rooms, and room track will be required?
4. What is the length of the average car haul?
5. What is the relative amount of power for ventilation?
6. What is the acreage of standing pillars, the estimated relative cost of production, and the estimated percentage of recovery?

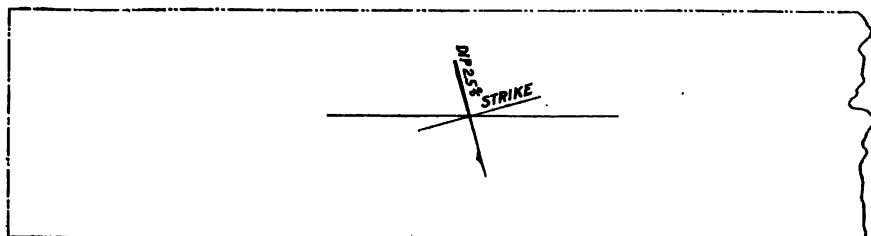


FIG. 5.—PLAN OF PROPERTY TO BE DEVELOPED.

It is not necessary to apply all of the methods of procedure to the development of the property under consideration in order to illustrate the thought in mind, and it has been decided to apply the methods referred to in Fig. 1, C, and the plans of mining in Figs. 3 and 4, as follows:

First Form.—Drive the third entry of the panel, turn the last two rooms on this entry first, start removing the pillar immediately upon the completion of the next to the last room, and continue to drive all of the rooms in the panel only fast enough to provide for the uninterrupted advance of the robbing. Work two men to the room and in the air courses and on the pillars. Only this method of procedure will be applied to the square and continuous panels, and the following methods to the square panel.

Second Form.—Drive the rooms of the panel as they are encountered, turning the first entry of the panel when it is reached, and start robbing immediately upon the completion of the last room on the third entry of the panel. Work one man to each working place.

Third Form.—Drive the rooms and entries of the panel as they are encountered, start robbing immediately upon the completion of the last room in the third entry, and continue the robbing until the completion of the panel. Work one man to every other room, but advance all rooms and work one man to each pillar.

Table II shows the comparison, as well as some other figures to which further reference will be made.

TABLE II

	First Form	Continuous Panel	Second Form	Third Form	Advancing Method
Output reached, months.....	53	42	62	92	7
Day laborers.....	60	65	82	102	31
	Advancing	method uses	8 asst. fore men.		
Mining machines..	9	9	14	18	8
Mine cars.....	275	310	335	465	155
Mules.....	18	22	24	32	10
Motors.....	4	4	5	6	2
Main entry.....	7,850	6,500	9,300	13,950	600
Main-entry track..					
Main-entry trolley					
Cross main entry	5,550	5,150	8,850	13,500	1,000
Cross-main-entry track.....					
Cross-main-entry trolley.....					
Room entry and..					
Room-entry track..	10,700	12,700	33,900	50,400	7,000
Room track.....	15,840	20,500	96,800	230,300	18,100
Average car haul...	6,180	5,333	7,420	10,230	3,640
Ventilation power, kilowatt-hours...	40	42	125	175	20
Acreeage of standing pillars.....	62.8	65.7	168.2	277.0	13.8
Relative cost of production.....	1.33	1.24	1.76	2.1	1
Percentage of recovery.....	94	95.5	83	80	97

From these data, it may be concluded that the first form of procedure and the plan of mining, Fig. 4, are the best.

The period of time required to reach the desired output was determined, for the several methods, as illustrated in Fig. 6, *A*; the location of the working faces from day to day, as determined by the assumed rate of advance of 16 tons per man, was plotted on a map, and the total number of faces at the time the desired output was reached were counted, from which data the tonnage curves were plotted as shown in Fig. 6, *B*.

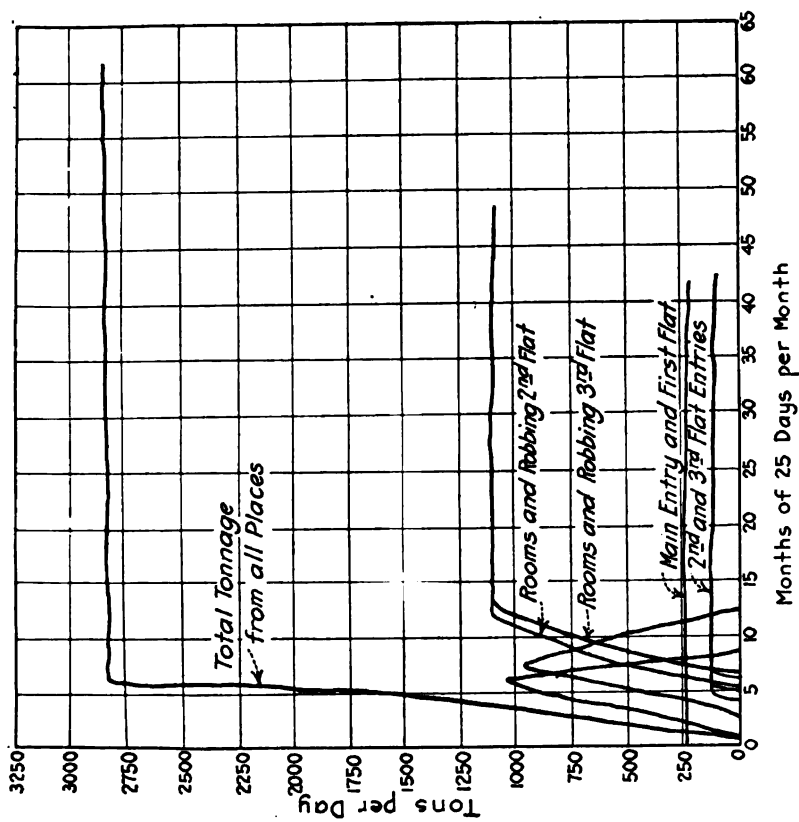


Fig. 6B1.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A1.

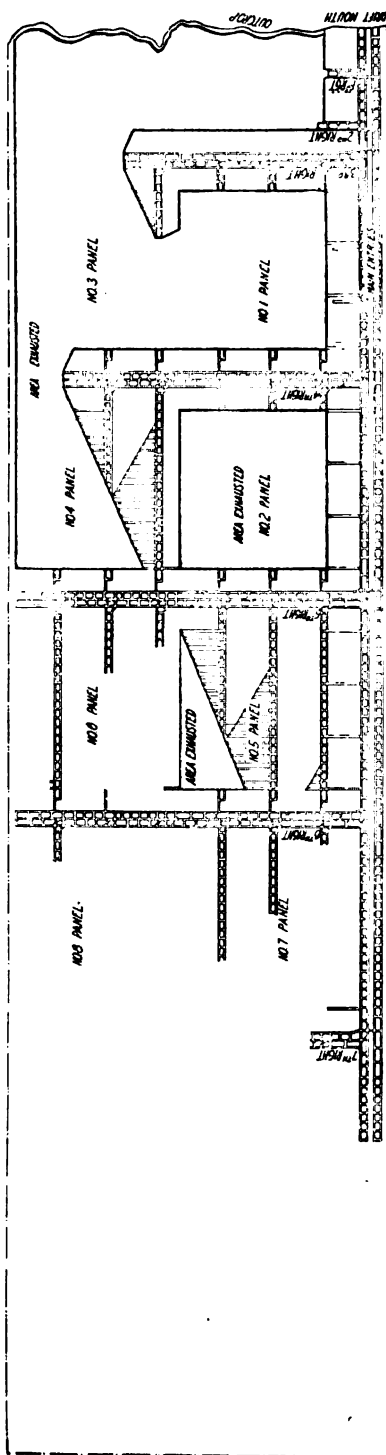


FIG. 6A2.—SQUARE PANEL WITH 2 MEN TO A ROOM, ROBBING RETREATING FOLLOWING IMMEDIATELY UPON THE COMPLETION OF THE ROOM. ROOMS 120 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

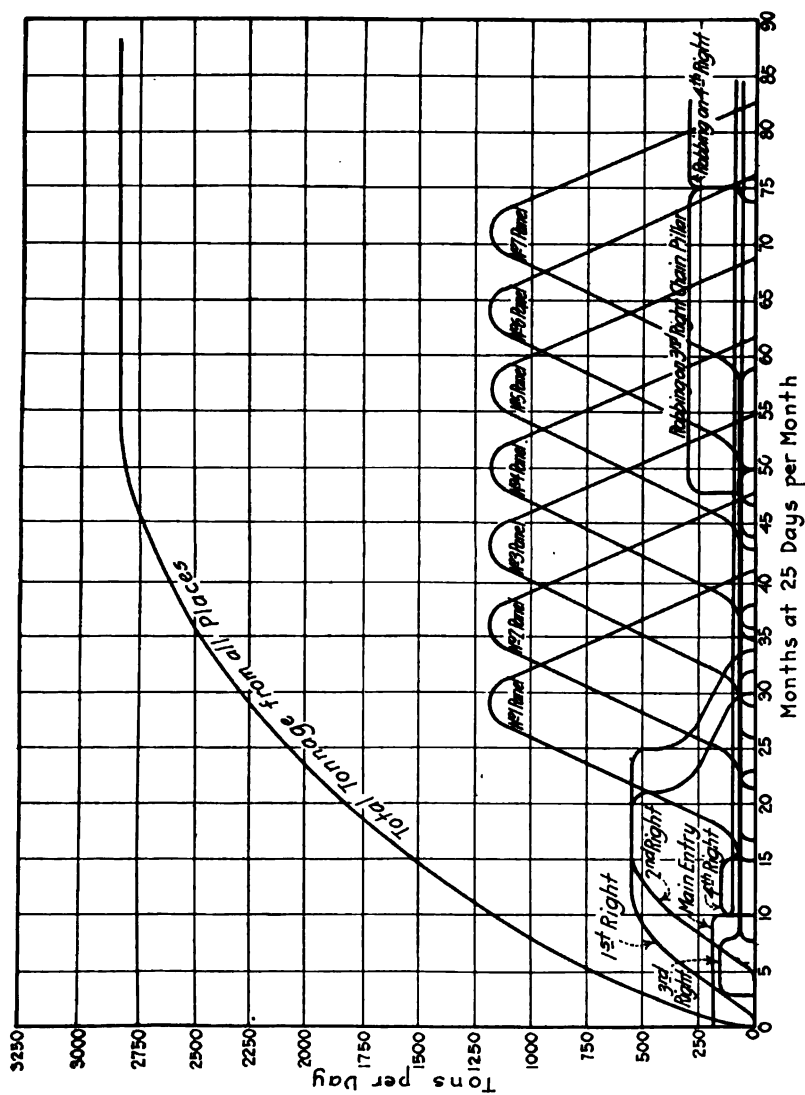


FIG. 6B2.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A2.

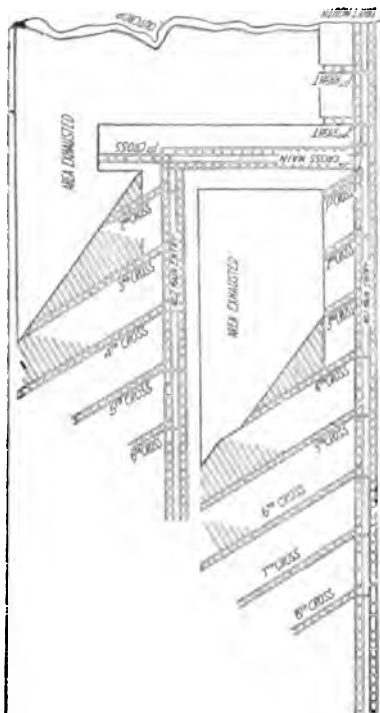


FIG. 6A3—CONTINUOUS PANEL WITH 2 MEN TO A ROOM, ROBBING RETREATING FOLLOWING IMMEDIATELY UPON THE COMPLETION OF THE ROOM. ROOMS 20 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

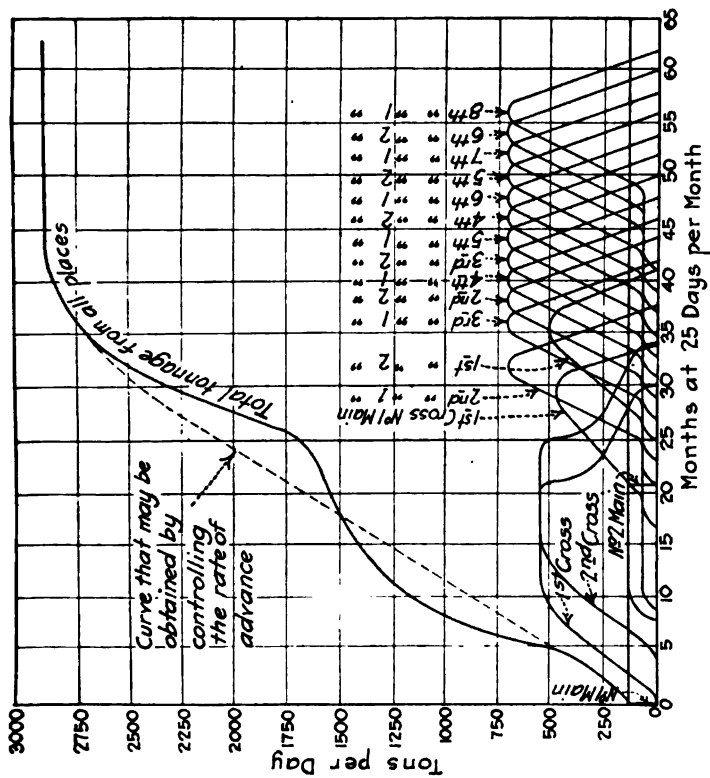


FIG. 6B3.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A3.

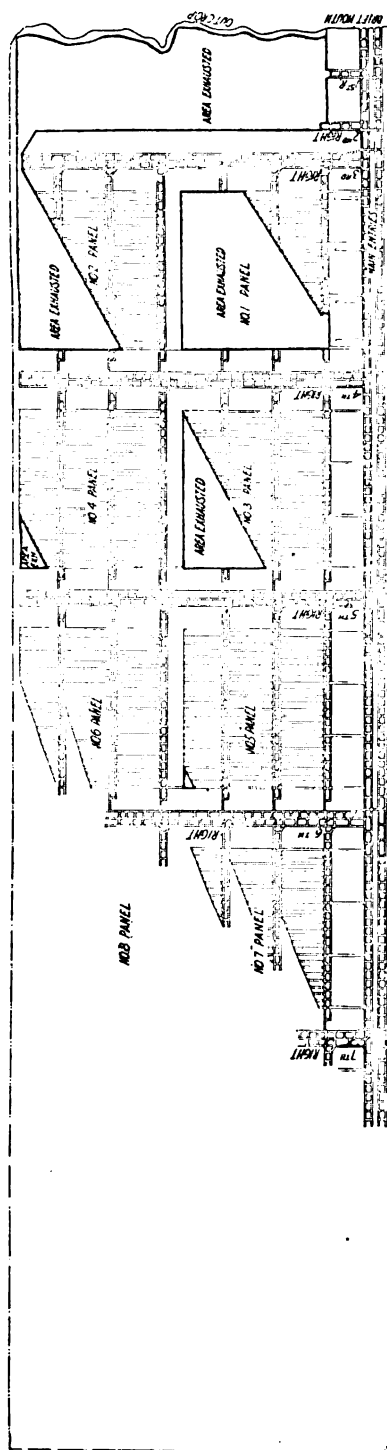


FIG. 6A4.—SQUARE PANEL WITH 1 MAN TO A ROOM. ROOMS ARE DRIVEN AS THEY ARE ENCOUNTERED AND ROBBING IS CONDUCTED RETREATING IMMEDIATELY UPON THE COMPLETION OF THE PANEL. ROOMS 20 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

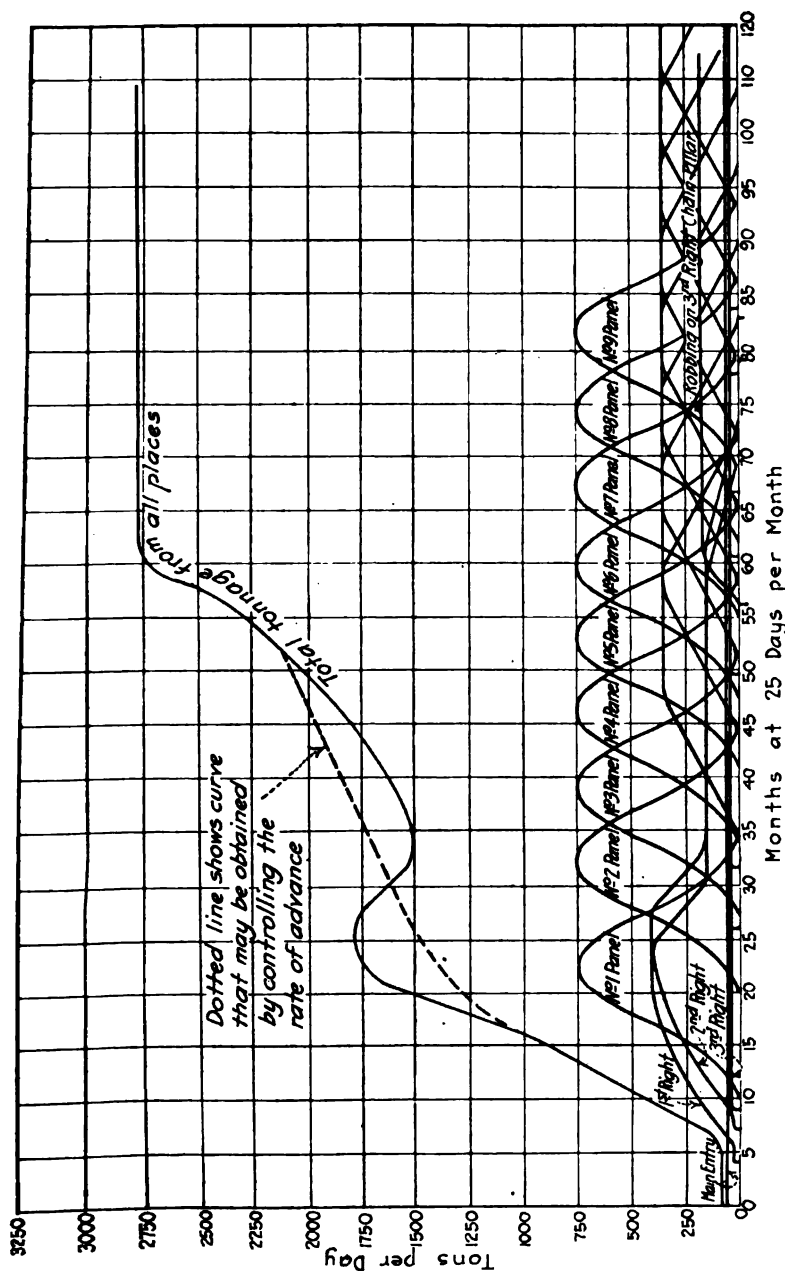


FIG. 6B4.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A4.



FIG. 6A5.—SQUARE PANEL WITH 2 ROOMS TO 1 MAN. ROOMS ARE DRIVEN AS THEY ARE ENCOUNTERED AND ROBBING IS CONDUCTED RETREATING IMMEDIATELY UPON THE COMPLETION OF THE PANEL. ROOMS 20 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

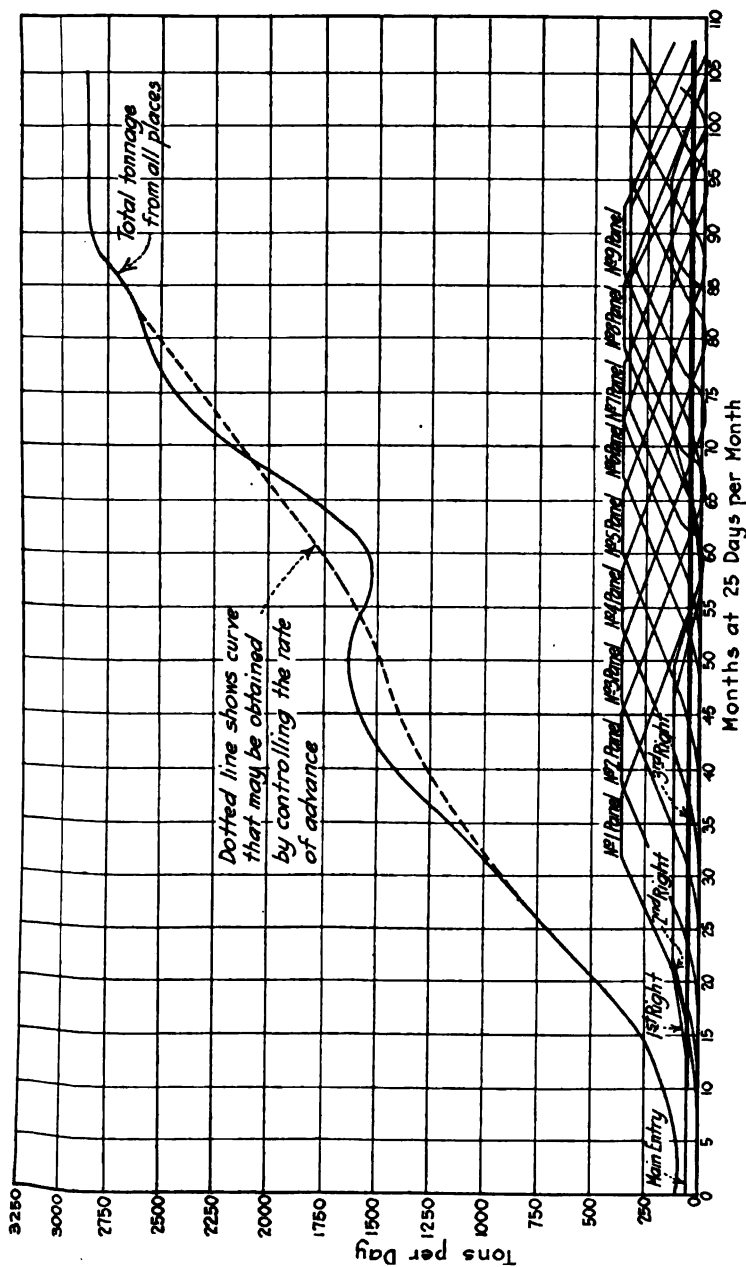


FIG. 6B5.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A5.

In arriving at the relative number of men and mules required, the writer used certain rates of performing certain tasks taken from time-study observations. The amount of rolling stock required is based on the assumption that the equipment will travel at the same rate of mileage per day; the other items compared were taken direct from the maps. In the absence of facts for comparison the writer used his opinion, based on observation, and the opinion of managers now operating under the plans compared.

All of the above methods of procedure and both plans of mining have been designed to meet certain wants. In some instances certain features of the plan have been prescribed by the land owners in order to safeguard their interests from "squeezes" and losses of coal due to lack of proper supervision. Were the proper supervision supplied and better methods of procedure adopted, the restrictions in the plan of mining might very properly be removed. Other details of design have been the result of accepting certain "rules of thumb" which have since been proved wrong, and yet other details, although admittedly wrong and expensive, have been introduced rather than combat the wrongs which they are designed to circumvent.

In the plan of mining shown in Fig. 3, the frequent interposition of barrier pillars is for the purpose of confining a squeeze and limiting its range of destructive action. The use of these barriers is imperative under the methods of procedure that involve large areas of long-standing pillars and where the degree of supervision is low. It is to be regretted that their use is so common, for they tend to interfere seriously with the maximum degree of concentration because one is seldom, if ever, able to provide a satisfactory output from a single panel, and then only for a short period of time. Where two or more panels are required to produce the output, the further the workings advance the more distantly separated they become, or other important considerations must be sacrificed. Disadvantages of the unit-panel plan may be seen in the curve in Fig. 7, which shows the great variation in the tonnage obtained daily, varying from zero at the opening of the panel, augmented by a more or less constant rate of increase, to a certain maximum number of tons, and then a gradual decline to zero again. If a certain number of tons per day gathered from the panel is accepted as 100 per cent. efficiency for a gathering motor, as shown in Fig. 7, it will be noticed that the motor is at first working at a very low efficiency, which gradually increases until the maximum is reached, at which time another motor must be added, and the average efficiency of the two motors is about 50 per cent.; there is a similar drop in efficiency with each motor that is added, until the maximum tonnage from the panel is reached, after which the process of removing motors from the panel is begun. In some measure this degree of efficiency may be increased by working the motors over more than one

panel, as is often done, and a better efficiency curve might be obtained more nearly in accordance with the full line shown, but in practice a rigid watch must be kept on this detail, or more often than otherwise a lower degree of efficiency than that shown will result.

If, in the preceding paragraph, instead of considering the efficiency of the gathering motor the efficiency of day laborers or the tons produced per unit of material and equipment in use had been considered, the same general discussion would apply. It is the writer's observation that where low efficiencies are obtained from day laborers, material, and

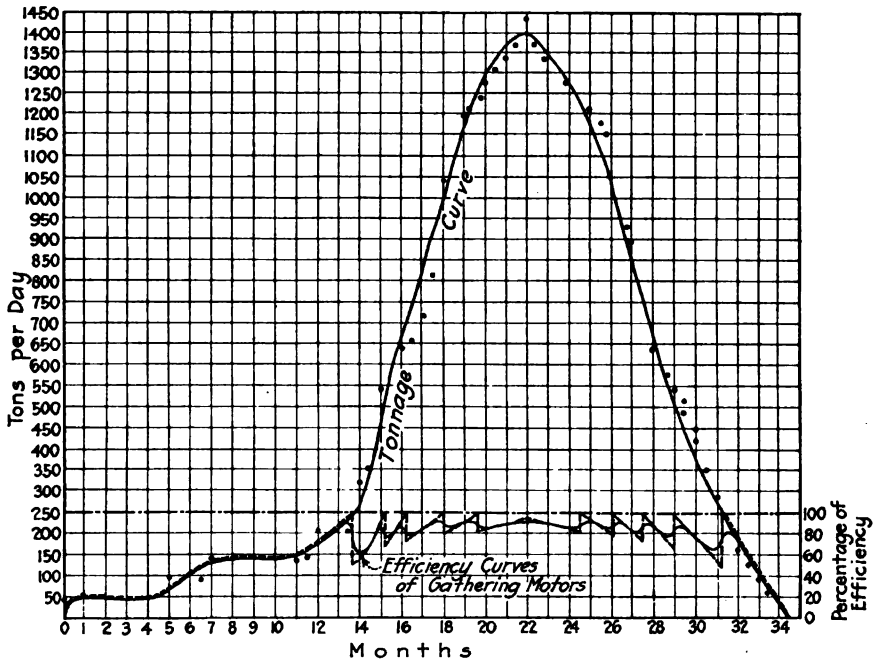


FIG. 7.—CURVES SHOWING THE VARIATION IN THE TONNAGE DAILY OBTAINABLE FROM THE UNIT PANEL WHEN PROCEEDING AS OUTLINED IN FIG. 6A2; ALSO SHOWING THE EFFICIENCY OF GATHERING MOTORS WORKING IN THE PANEL.

equipment, low efficiencies are also obtained from the miners at the room face. For these reasons it is difficult, and in practice well-nigh impossible, to establish any constant relation between a given tonnage desired to be uniformly produced, and the amount of material, equipment, and day laborers required to produce that tonnage; the efficiency of these quantities rises and falls with the rise and fall of the tonnage curve, although in an erratic manner.

Thus it would appear that the square panel, while designed to meet certain requirements, does so at the loss of much that is to be desired, and introduces new complications. The barrier pillars are, as the

term implies, for the purpose of barricading against some impending danger, such as an unforeseen squeeze. Since no one can predetermine where or when these squeezes will occur it sometimes happens that barrier pillars are provided where they are not needed, and are omitted where they are needed; yet experience has shown the wisdom and necessity of their use under certain conditions. They would be used less frequently if the square panels were made rectangular, but the same degree of security would not be obtained unless the entries were driven to the limit of the rectangle, with few or no rooms driven as the entries advance. If we accept it as axomatic that when a room is driven to completion its pillars should be immediately removed in order to obtain the best results, or that it is equally as fundamental to open up no new entries until ready to mine from them, and that mining should then be conducted at the maximum rate of production, the rectangular panel that involves either long-standing pillars or long-unproductive entries must be rejected.

The continuous panel obviates the necessity for frequently interposing a barrier pillar and it is especially well adapted to a property where the main entries are driven to the dip. However, the tonnage from a single continuous panel is limited, and where the main entries of the property go to the rise the maximum degree of concentration cannot be obtained or the rooms off of the cross entry will go to the dip. Advancing robbing is impracticable because the pockets in the pillars go to the dip. The rate of production from a single room entry rises and falls in the same manner as the rate of production in the room entries of the square panel and the general discussion above in reference to the square panel applies to the continuous panel.

However, if one follows the history of the development of mining methods from the early-day single-entry system to the present-day panel system, it will be found that the square or nearly square panel meets sound mining practice more closely than any of the plans which have preceded it. Until methods of procedure are adopted which make the restrictions of the panel unnecessary, or until a plan of mining is devised without the objectionable features of the panel, but retaining its many favorable features, the square panel will be accepted by many operators as the standard plan of mining.

For many years it has been the common belief that coal could be most economically cut and blasted by using a depth of cut equal to the height of seam. This erroneous idea frequently resulted in blasting down more coal than could be loaded in one day and not enough for two days or for two men in one day, and was the cause of allotting more than one room to a miner. That the height of seam does not bear any direct relation to economical cutting or blasting was demonstrated by the United States Coal & Coke Co. at Gary, W. Va., working with a Sullivan

shortwall machine, and the writer's observations are that miners are pleased to work two or more to a room, provided their earnings are as great as when they work in rooms by themselves.

Much thought is being given to the subject of mining methods. Probably the most marked results have been achieved by the officials of the above-mentioned company. They realized the objections to the mining methods outlined above and applied themselves to working out a plan which would be simple, direct, and efficient. They accepted it as axomatic that any change in the prevailing plans of mining must be beneficial to the property owner, operator, and miner alike, for any change that would benefit one or more of the interested parties at the expense of the others would not last.

In this study difficulty was experienced because of the entire lack of systematized knowledge as to the proper relative rate of advance of room to retreat of pillar, the most economical width of room, and in fact what might be considered 100 per cent. efficiency for any man, animal, or machine about the mines. In order to determine these data, which were absolutely essential to an intelligent solution of the problem, a series of time studies was instituted and extended over a period of weeks, covering all of the motions that make up certain underground operations that have to do with getting the coal from the working face to the railroad car. Thousands of observations were taken, properly checked, tabulated, collated, and used as a basis for a method of procedure, which has been put to the rigid test of practical use with remarkably good results.

This method of procedure has for its object the maximum degree of safety, sanitation, and opportunity to the miner, and of security to the property owner, while at the same time offering the greatest advantage to the operator. It combines a maximum degree of concentration with a minimum of expenditure for labor, material, and equipment, in such a manner that these *quantities bear a constant relation to the output*. Its use has resulted in a marked reduction in fatalities, increased earnings to the miners, decreased costs per ton for labor, material, equipment, and capital, and the recovery of practically all the coal in the seam.

At Gary, W. Va., mules are used for gathering, and as a result of concentration their efficiency has, in some instances, been increased over 200 per cent. At one of the mines, fewer day laborers are employed underground than are employed about the tippie.

For the purpose of comparing the results obtained under this method with those from the several methods of procedure in the panel system, the writer has applied the method to the property and the problem under consideration. Fig. 8 shows the arrangement of the workings at the time the desired maximum output is reached. It also shows the details of the method of procedure; the other data desired are given in

Table II. Fig. 9 shows the tonnage curve and, for comparison, the total tonnage curve from Fig. 6B2. The tonnage curve, Fig. 9, from the unit entry shows that the tonnage rises very rapidly until the maximum is reached and then continues indefinitely at that rate of production. By using available data, the proper length of room, angle of breakline and angle of advancing faces may be predetermined, so that the total daily tonnage from the entry is a multiple of the tons that can be hauled

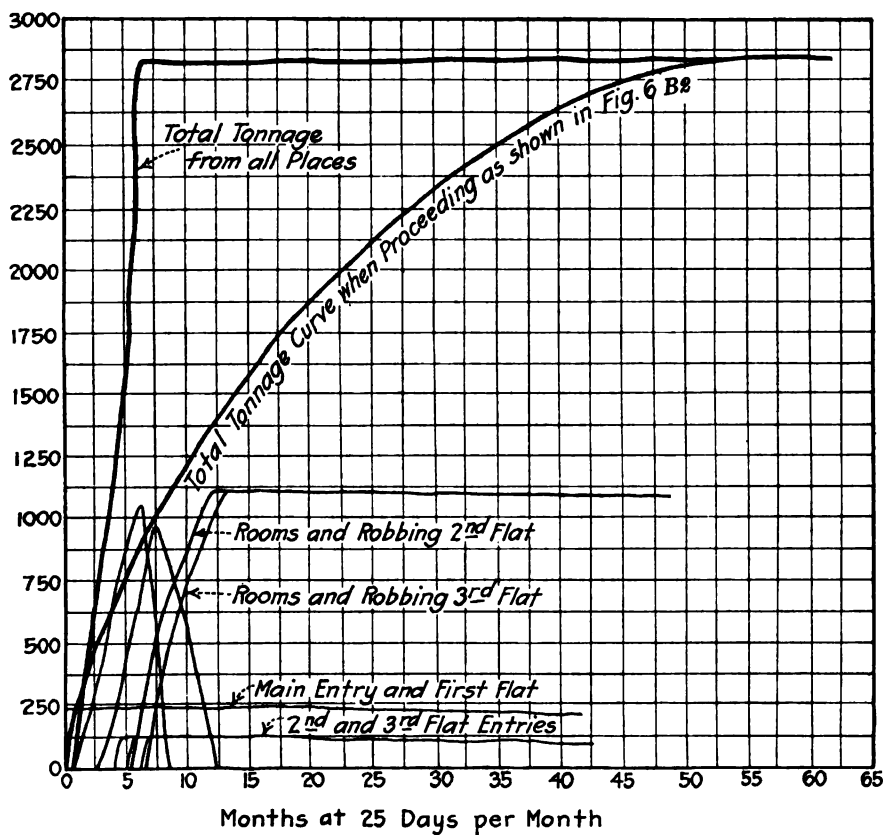
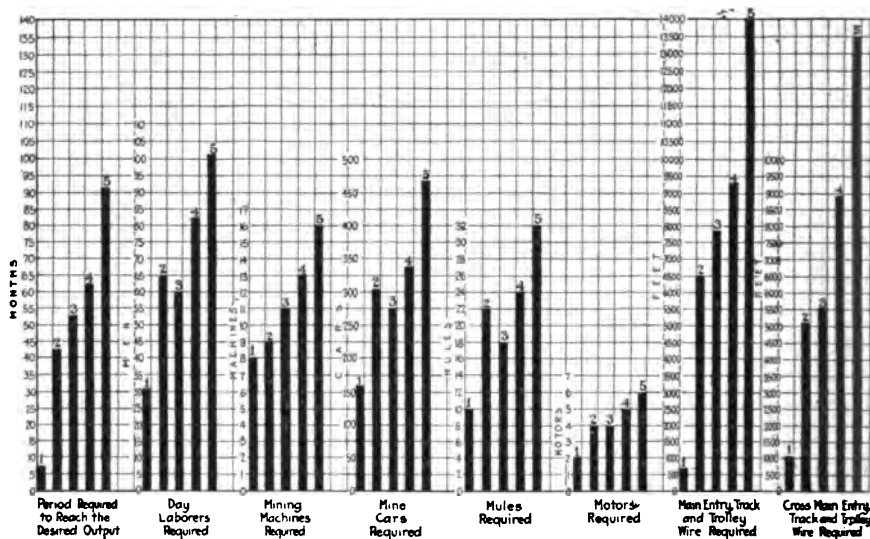


FIG. 9.—CURVE SHOWING THE RATE OF DEVELOPMENT TO THE DESIRED OUTPUT, UNDER THE METHOD OF PROCEDURE, SKETCH F, FIG. 1, AND THE ADVANCING PLAN OF MINING, FIG. 8. ESTIMATED PERCENTAGE OF RECOVERY, 97 PER CENT.

by a mule or motor; thus, the mules or motors are always working at maximum efficiency. It is equally true that when the workings have advanced for a short distance, after reaching the maximum tonnage from the entries, the estimated minimum number of day laborers required may readily be confirmed, and once the entry reaches its maximum tonnage, and the quantities of labor, material, and equipment have

been accurately determined, these quantities remain constant throughout the entire extent of the entry, which may be as great as the property is long.

Fig. 2 shows that the room space occupied per miner is less than in any of the other methods now in effect, which is an index of the relative degree of safety a miner obtains for a given expenditure of time and energy. The excellent manner in which the rooms are timbered, shown in Fig. 8, is the minimum required; where the mine foreman or miner has reason to believe that additional timber is required to make the



1. Advancing system with 4 men to a room.

2. Continuous panel with 2 men to a room, robbing retreating following immediately upon the completion of the room.

3. Square panel with 2 men to a room, robbing retreating following immediately upon the completion of the room.

4. Rooms 36 ft. wide, 54 ft. of pillar.

FIG. 10.—GRAPHICAL COMPARISON OF THE AMOUNT OF LABOR, MATERIAL, AND PROPERTY SHOWN IN FIG. 5 WHEN FOLLOWING THE METHODS OF PROCEDURE AND THE RELATIVE COST PER TON, THE RECOVERY, AND THE PERIOD OF TIME REQUIRED

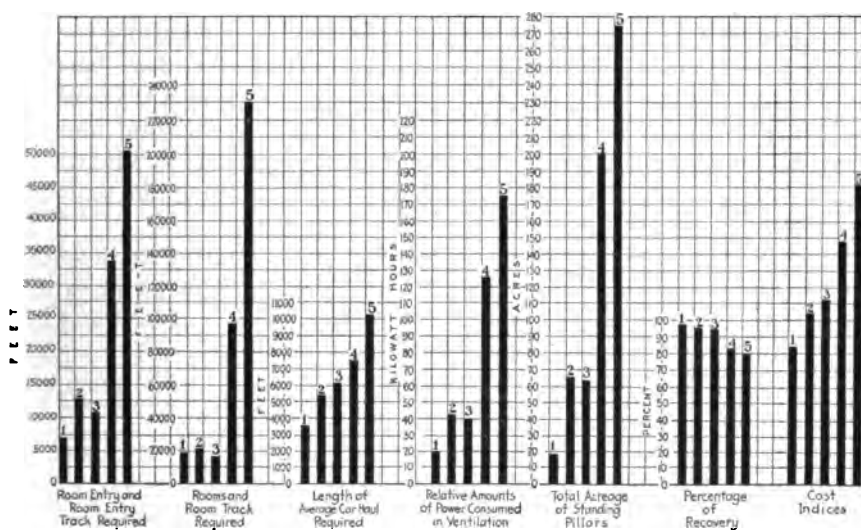
place safe, the miner must place additional timber before doing anything else. That these precautions and the high degree of supervision exercised are worth while may be seen from the paper by Howard N. Eavenson.¹

As the entries advance, all rooms are driven and robbed immediately upon their completion, and rooms are opened up only fast enough to provide for the uninterrupted advance of the robbing. Thus no barrier pillars are required, for the virgin coal protects the workings on three

¹ Safety Methods and Organization of United States Coal & Coke Co., *Bulletin*, No. 98, February, 1915, pp. 413 to 430.

sides and the weight of the roof is resting on the bottom in the robbing. If a disturbed area of coal is encountered, or for some reason it is desired to discontinue the panel, a barrier pillar may be introduced at any time exactly where it is needed and the entries continued for the purpose of exploration.

In order that the different methods of mining may be readily compared, Fig. 10 was prepared showing the relative amount of labor, material, and equipment required to produce the tonnage desired from the property shown in Fig. 5; also the acreage of standing pillars, the relative cost of production, and the estimated percentage of recovery.



4. Square panel with 1 man to a room. Rooms are driven as they are encountered and robbing is conducted retreating immediately upon the completion of the panel.

5. Square panel with 2 rooms to 1 man. Rooms are driven as they are encountered and robbing is conducted retreating immediately upon the completion of the panel.

2, 3, 4, 5. Rooms 20 ft. Wide, 40 ft. of Pillar.

EQUIPMENT REQUIRED TO PRODUCE AN OUTPUT OF 2,800 TONS PER DAY FROM THE PLANS OF MINING OUTLINED ABOVE. ALSO THE ACREAGE OF STANDING PILLARS, TO REACH THE OUTPUT.

The writer has given much time and study to the subject of this paper and concludes that any method of procedure that does not provide for the removal of pillars immediately upon the completion of a room is fundamentally wrong, because it involves long-standing pillars open to the unfavorable influence of atmospheric agencies and other forces of nature; the duplication of track work; the cleaning up of many slate falls that might otherwise have been avoided; and the scattering of workings, all of which increase the cost per ton for labor, material, and equipment, and cause the pillar coal to be badly disintegrated and low in domestic and lump sizes.

It sometimes happens in practice, however, that fundamentals must be sacrificed to adapt the method to peculiar conditions encountered, often resulting in lack of concentration and large areas of standing pillars. Where considerable tonnage is desired and a new property is being opened, skilled miners, experienced in robbing pillars, are hard to get and frequently the officials, mine foreman, and underbosses are not experienced. In order to keep up the tonnage under these conditions, the workings must necessarily become distantly separated, because coal can only be obtained from room workings. It frequently happens also that the rates for mining pillar coal and room coal are not properly adjusted, so that the men can earn more in room work than in pillar work, naturally causing the pillars to lag behind, and requiring the introduction of barrier pillars to safeguard against squeezes; these barriers in turn cause a further separation of the workings, and a decrease in the efficiency of labor, material, and equipment. The natural impulse of the mine foreman, therefore, is to open up more rooms in advance of the robbing in order to increase the efficiency to something like a proper standard.

For these reasons the territory for a given output during the development period should be as isolated as possible, and no greater in extent than is practicable. After the development period is passed and the organization perfected, in the opinion of the writer, there is no good reason why a mine operation should not be conducted with much the same regularity as a blast furnace or an industrial railroad.

The fallacy that the average miner will load only so much coal and no more has long since been exploded, and it is a matter of every-day observation that miners are pleased to load coal if the mine cars are given to them with some degree of regularity and with some relation to the time required to load a car. Not long since the writer observed the tally sheets at one of the mines visited, and was pleased to note that of 132 miners loading that day, the average tons per miner per day was over 250 per cent. more than the average of the State. When one considers, however, that a coke loader, working under the heat of the sun and of the coke ovens, will load from 35 to 40 tons as an ordinary day's work, there is no reason why a miner working under so much more favorable circumstances should not load at the same rate. In this connection the following observations that have to do with loading coal underground are interesting.

These figures show that less than 47 per cent. of the time spent underground was consumed in loading coal and over 12 per cent. of the time was lost waiting for the empty mine cars. It may be stated further that these men were loading at the rate of 35 tons per day of 8 hr., and actually did load at the rate of 16 tons per man per day per year.

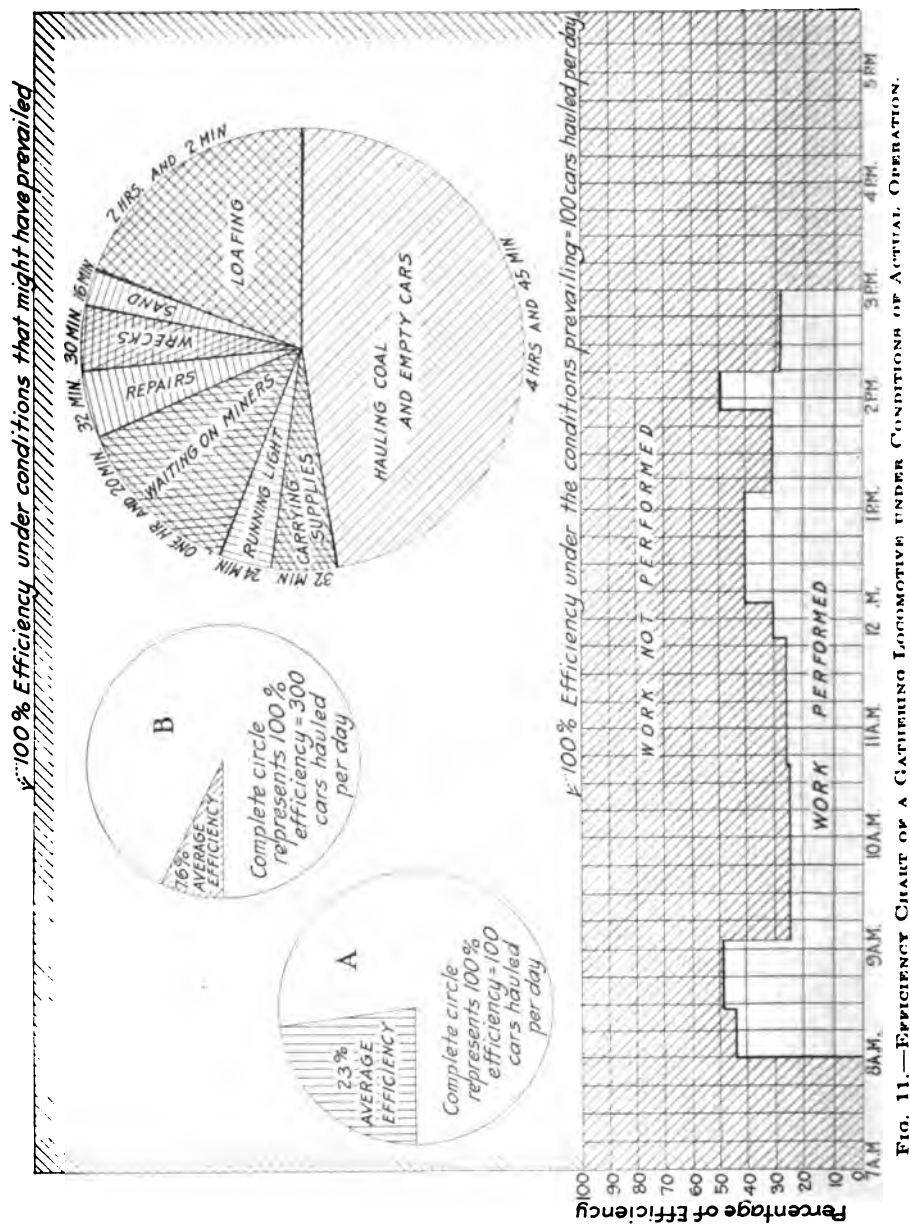
Thickness of Seam, Feet	Loading Coal, Minutes	Waiting on Cars, Minutes	Resting, Minutes	Length of Shift, Hr. Min.
5.50	220	58	10	7 42
4.67	193	60	13	7 16
6.17	221	65	13	7 36
6.58	221	65	13	7 36
6.33	220	57	10	7 35
6.17	220	57	10	8 09
6.00	221	65	13	8 16
6.92	221	65	13	9 00
5.33	221	65	13	7 57
6.00	198	60	13	7 02
5.00	193	55	10	7 04
8.08	231	32	13	7 06
Average.....	214	59	12	7 40

The results obtained by the United States Coal & Coke Co. under the advancing system as compared with those obtained under the several other methods of procedure noted above may seem unreasonable and beyond expectation, but they fulfill the anticipations based on the data available when the method of procedure was formulated. Better results, in industries generally, invariably have been found to follow the presentation of opportunities to workmen, concentration, regularity, and the elimination of what may be termed lost motion, as shown by the marvelous increases in efficiency obtained by Gilbreth, Taylor, and other pioneers following the same line of thought. All of us remember how Gayley startled the metallurgical world by the introduction of dry-air blast, eliminating the periods of lost motion, which permitted of regularity and concentration in the operation of the blast furnace and produced results which at that time were almost beyond human credulity.

But in every instance where high records of efficiency have been obtained, it should be particularly noticed that the foreman and workmen were carefully following a plan, all the details of which had been carefully worked out by the management.

A detailed plan of mining and a projection with written instructions should be worked out by the management for every mine, and the mine foreman and laborers should not be permitted to deviate from same without the written consent of the management.

Curves and graphs should be plotted, showing an adopted 100 per cent. efficiency for the men and equipment, and curves should be plotted on the same chart showing what is actually being accomplished, as illustrated in Fig. 11, which is a time chart of a gathering locomotive under working conditions found in the mine, the summation of which is given in Fig. 11, A.



Whether or not it is practicable for operators working under any one system of mining, or any method of procedure with a given system of mining, to introduce another system or method of mining, the writer does not presume to discuss. The assumption is, of course, that each operator is doing what he considers best.

It might be inferred, however, from the example mentioned above that concentration and motion studies can be applied only where changes in the plans of mining are effected, or for the purpose of showing inefficiency. Many examples of constructive results might be mentioned and it is desired to call attention to Fig. 12, which is a reproduction of a mine now operating. It may be noticed that the mine is progressing

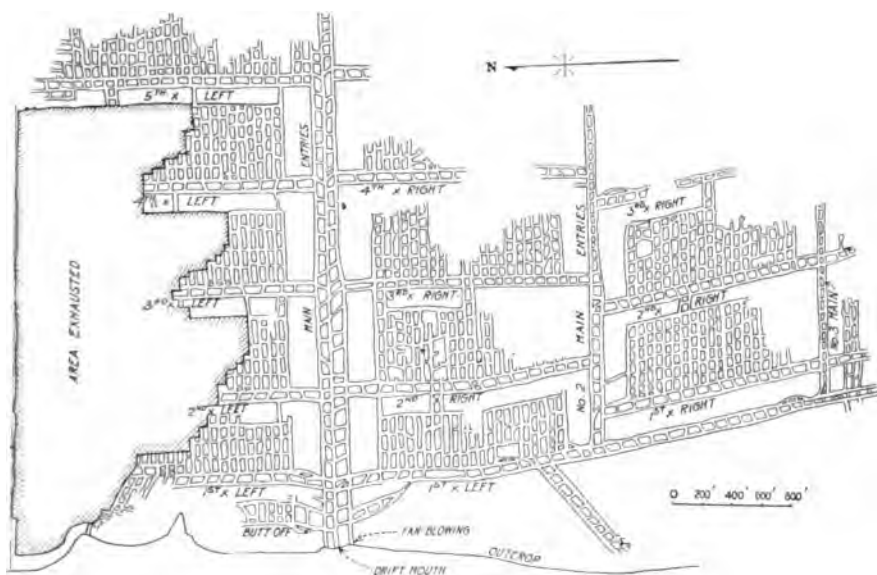


FIG. 12.

on a rectangular panel system, with workings off of No. 1, 2, and 3 main entries. Low efficiencies were obtained throughout the mine as a result of the widely separated condition of the workings and it was concluded to suspend operations on the No. 2 and 3 main entries, concentrating the men in the 1, 2, 3, and 4 left cross entries. It is emphasized that no change in the plan of mining was involved, but concentration, effected practically over night, resulted in a 50 per cent. decrease in the main-line motors, 47 per cent. decrease in the day laborers, and 30 per cent. decrease in the number of mules. The daily output was increased to the highest figure in the history of the mine and the daily earnings of the miners were also increased.

Changes in the method of procedure and plan of mining are now being

made, *without involving capital expenditures*, from which it is anticipated a very much greater degree of concentration will be effected, resulting in decreased cost of production, increased percentage of recovery and higher daily earnings to the miners, while at the same time reducing the amount of labor, material, and equipment in use.

The purpose of this paper will have been accomplished if it arouses an interest in the study of methods of procedure in their relation to the cost of production, and tends in the slightest degree toward safety and the conservation of human lives and coal resources.

In conclusion, the writer desires to thank Thomas H. Clagett, Chief Engineer of the Pocahontas Coal & Coke Co.; Edward O'Toole, General Superintendent, and Howard N. Eavenson, Chief Engineer of the United States Coal & Coke Co.; and James Elwood Jones, General Manager, Pocahontas Consolidated Collieries Co., Inc., for data and criticism in the preparation of this paper; and Warren T. Russell, Assistant Mine Inspector of the Pocahontas Coal & Coke Co., for assistance in the preparation of the figures.

DISCUSSION

S. A. TAYLOR, Pittsburgh, Pa.—In connection with the conditions that exist in the Pocahontas field, is not the question of lump coal less of an item in the sale of the product than it is in some other sections? Take for instance the Pittsburgh section, we would be embarrassed considerably by the use of what you have termed the continuous panel, from the fact that it does not take into consideration the cleavage of the coal, and our output would contain a greater amount of fine coal, when operating under the continuous panel than under the square panel.

MR. GRADY.—In the Pocahontas field cleavage is not as highly developed as in some other sections of the country; for that reason it does not exert much of an influence in laying out the mine projection. It is also true, as you say, that the question of lump and prepared grades of coal has only recently exerted an important influence in the Pocahontas field. But to-day everybody is striving for a higher percentage of lump and prepared sizes because of the new markets, and the uses to which Pocahontas coal has been put, and the decreased demand for beehive coke.

It is generally admitted to be the better practice to take advantage of the cleavage, but if in attempting to do so it becomes necessary to drive workings to the dip or conduct robbing up hill, it is then a question whether or not the increased percentage of prepared sizes is worth more than it costs; if so, then undoubtedly it is proper to drive to the dip and rob up hill. If, however, you are able to drive the rooms on the faces, you will notice that only the faces of the main entries are adversely located, and

with machine cutting, that is not a very serious disadvantage as compared with the advantage of having the grades in your favor.

But generally speaking each mine is a problem in itself and just what is the best thing to do can only be determined by a careful study of the mine.

GEORGE S. RICE, Pittsburgh, Pa.—Fig. 3 indicates a system, but I see no indication of how the chain pillars and entry barrier pillars are withdrawn. There seems to be a very large amount of coal in the chain and barrier pillars.

MR. GRADY.—The chain and barrier pillars are taken out when it becomes no longer necessary to hold the entry open as a haulway. The entries as shown in Fig. 3 are in use as haulways, but in Fig. 6A2 you will notice that in some instances the panels are completed and the chain and barrier pillars are being extracted.

H. M. CRANKSHAW, Lansford, Pa.—What is the best percentage of extraction in advance work before robbing is started?

MR. GRADY.—The percentage of extraction, or the ratio of the width of the room to its pillar, in first workings depends on many factors and in general it will be found that the economical use of labor and material is the limiting factor, rather than stratigraphy. The amount of superincumbent strata, and their nature, the thickness of the seam and its nature, the nature of the immediate top and bottom; what constitutes the conditions under which a miner may work most economically; the economical use of labor and material; how long the pillars may stand, and what falls if any may be expected in the rooms during the period intervening between the first workings and the robbing—these are some of the more important factors to be considered in determining the proper ratio of room and room pillar. Probably as important a factor as any is the nature of the stratum immediately overlying the seam, or the "nether roof," as it is called. If this stratum is either very weak, say 30 in. of draw slate that falls on exposure to the air, or very strong, say gray sandstone, larger pillars will be required than otherwise in order to provide for the proper recovery of the pillar.

Where high recoveries and low cost are obtained, the present practice is to use room widths one-fourth to two-fifths of the distance from center to center of the rooms. Many mines have come under my observation where the room width was much greater; the costs were high and the percentage of recovery was from 50 to 75 per cent. A table has been prepared by the writer showing the limits beyond which the ratio of room width and pillar width must not go, for rooms 18 to 40 ft. wide, widths varying at intervals of 4 ft., and for covers from 100 to 1,200 ft. thick, the thickness varying at intervals of 100 ft.; which might have been introduced as part of this

paper, because the question enters into the cost of production, but it was concluded not to burden further the discussion.

MR. CRANKSHAW.—As far as concentration is concerned, do you consider instances of two men employed in one room?

MR. GRADY.—Concentration may be effected in almost every detail of mine work, but where two or more men are employed in a room a greater degree of concentration is possible. It is considered good practice to work as many men in a room as controlling factors will permit.

MR. CRANKSHAW.—At a mine I have in mind, the coal was about $8\frac{1}{2}$ ft. thick. Two men were worked per room, the size of the cut and the width of the room being so arranged that about 40 tons per day were shot down. The shooting was done after hours. This allowed about 20 tons per day per man. Do you know of any better conditions of work than this?

MR. GRADY.—Twenty tons per man per day has been the average output for several years right along.

MR. CRANKSHAW.—These figures were taken for a period of three months.

MR. GRADY.—Was that the average output of the entire mine?

MR. CRANKSHAW.—No, I am referring to one particular panel.

MR. GRADY.—The figure that I have used—*i.e.*, 16 tons per man—is conservative. It is a figure obtained not for one particular section of a mine, or for one month, but has been obtained over a period of time. I find that the average of several mines, which come under my observation, was about 16.4 tons per miner per day, for the year 1914. Much depends upon the opportunity presented to the men; you will find practically everywhere that men are loading at a rate of from 30 to 40 tons per day, and I have no doubt but that miners could load 25 tons per day of 7 hr. if the opportunity were presented to them and the conditions were suitable. To-day little attention is paid to the matter of fatigue or to some of the most simple, though inexorable, laws of nature that have to do with a man's performance.

MR. CRANKSHAW.—Are mules being used for hauling the cars?

MR. GRADY.—Yes.

MR. CRANKSHAW.—Is it your opinion that it is more economical to use the mule for haulage where the grades are favorable?

MR. GRADY.—The question of mule haulage vs. motor haulage is an old one. There are conditions about some mines that preclude the possibilities of economical mule haulage and in all mines there are conditions where the motor will do from 10 to 40 times the work of a mule; but likewise in many mines there are conditions where for the same capital expendi-

ture for mules and the same operating expenditure for labor and material, the cost per ton for gathering will be less with mules than with motors.

The reason is obvious; under favorable conditions mules will travel under load all day at the rate of about 2.5 miles per hour, whereas under the same conditions motors will travel at the rate of about 5 miles per hour, but in handling the same load as the mule under the same conditions the motor cannot be under load more than half the time. It takes two fairly high-priced men to operate a motor and one fairly low-priced man to drive a mule, and the cost of cable alone sometimes equals the cost of maintaining the mule. The capital expenditure for the motor is about 10 times that of the mule. But unfortunately in practice sufficient attention is not paid to differentiating between mules and motors. Often you will find mules working where motors should be, and *vice versa*.

MR. RICE.—You speak of recovering 97 per cent. Is that theoretical or is it an actual figure?

MR. GRADY.—The problem under consideration is a hypothetical one, and the figure to which you refer is an estimate, but it is based on figures taken from actual practice. Not long since I had occasion to figure the percentage of recovery from about 780 acres that had been mined, over 300 acres of which had been final mining, and the average percentage of recovery was above 94 per cent.

MR. RICE.—The percentage of recovery depends, of course, upon what you estimate to be the thickness of the seam. I do not see how you could get so complete a recovery with any system; you lose some pillars, do you not?

MR. GRADY.—The method used to calculate the percentage of recovery is as follows: Accurate surveys are made, not to exceed 90 days apart, from which the area is determined by plotting and planimentering same. Sections of the seam are taken at such intervals as are necessary to obtain an accurate average section, and the volume is calculated by multiplying the average section by the area. Usually the total acreage exhausted in any mine will be divided up into small units of about 1 acre and the error between the actual volume excavated and the calculated volume will be small. The tonnage of these volumes is readily determined by applying the proper constants and the total tonnage, divided into the actual tonnage delivered from the mine, as recorded at the mine mouth, gives the percentage of recovery.

A method of calculation for determining the percentage of recovery sometimes used is the ratio of the acreage mined divided by the acreage mined and lost. This method is obviously wrong because of the varying section, but it could not be used in many mines of the Pocahontas field because the area lost cannot be measured in acres, because: (1) the area lost is seldom in excess of a few square feet in any one loss, and (2) there

is no one there to keep account of the area and the number of losses at the time the loss occurs.

MR. RICE.—That is a very good showing for any mine. In my own experience, even with longwall mining, I was not able to get more than 97 or 98 per cent., though that somewhat depended on what you estimate or assume to be the thickness of the vein. My observation down in the Pocahontas field is that some coal is lost by sticking to the rejected bone coal, besides what may be lost through crushing or covering up in pillar withdrawal.

MR. GRADY.—In my opinion percentages of recovery are not absolute. But if you apply the same method of calculation to 75 or 100 properties and you find that some are obtaining 75 per cent. recovery, while others are obtaining 95 per cent., you are justified in saying that one is obtaining a 20 per cent. higher recovery than the other. Or if you are making these calculations from year to year in connection with a given property you are able to make fairly accurate comparisons.

EDWIN LUDLOW, Lansford, Pa.—In the advancing system of mining to which you refer, do you use no panels?

MR. GRADY.—The policy is to open up from one to three entries, as the condition of the mine will permit, sufficient to give the maximum output desired (varying proportions of the maximum output may be obtained by changing the angularity of the robbing lines and the lines of advancing faces, or the number of men worked per room). Rooms are opened up only fast enough to provide for the uninterrupted advance of the robbing, so that you have good robbing falls behind, virgin coal on three sides, and a very limited acreage of active pillars in the immediate vicinity. Thus the possibilities of a squeeze are most remote and in case one should start it can readily be controlled.

Some of the advantages claimed for panels are that in the event of a catastrophe its range of destructive action is limited. This is wrong from two points of view. Catastrophes will always be with us so long as the line of defense is to limit their range of destructive action, rather than to eradicate their cause, and as a matter of fact, panels do not limit their range of destructive action. However, gas and dust accumulate in standing pillars, and by referring to Fig. 10 the relative possibilities for accumulations of gas or dust in the advancing system as compared with other systems may be seen.

Panels introduce numerous barrier pillars that might otherwise be avoided, and from the analyses in the paper it appears that their elimination is very much to be desired.

MR. LUDLOW.—You have a main haulage entry, do you not? What barrier pillars are provided there?

MR. GRADY.—In the example shown in Fig. 8 the main haulage is through 1st flat; 1st, 2d, and 3d flats are continued to the extreme inside of the property. The solid coal flanking 1st flat, while serving in the capacity of a barrier pillar, is really acreage held in reserve, so that at the time the chain pillar of 1st flat is removed there will be sufficient coal tributary to the entry to provide a tonnage that may be readily mined without experiencing an increase in the cost of production or a decrease in the output.

MR. RICE.—There is one phase I am interested in inquiring about. I have not been in the Pocahontas field for several years, except for the investigation of several unfortunate disasters, but several years ago I paid a good deal of attention, while carrying on extensive mine sampling with a party in the Pocahontas field, to the matter of waste. I observed that under the systems then in vogue there was a considerable difference in the quality of the coal as loaded, between that from the faces and that from the pillars, apparently for this reason: The bone coal is easily rejected in loading in the advance workings; but in the older pillars the layer of bone, which is next the roof, was crushed down and so mixed with the coal that it was not possible for the miner to separate or pick it out. For this reason I imagine that it is an advantage to take out the pillars immediately.

MR. GRADY.—That is very true, in my opinion. I do not have comparative figures over a long period of time, but I am reliably informed that there is a very marked increase in the percentage of prepared sizes, and at one place, where there is a washer working, an increase in the car yield is reported, due, I take it, to the decrease in the amount of slate mixed in with the pillar coal.

Pillars I have no doubt deteriorate some if they stand for years, and in many instances pillars are being mined to-day that are from 5 to 8 years old.

MR. RICE.—It was so in the case I spoke of. Do you not lose some of the curtain wall, or shell of the pillar next to the gob or robbing fall?

MR. GRADY.—Yes, some of that shell is lost in many instances. If the robbing falls are not following closely upon the removal of the stump of the pillar, and the rooms are relative free from waste, the quantity of coal lost in the shell is very small; but where robbing falls follow quickly and the rooms are dirty many people use coal as a protection, and in order to avoid loading up slate. A better and cheaper practice is to set a row of protecting timbers.

The point you have brought up emphasizes the value of wide rooms and wide pillars, where conditions will permit of their use. You have fewer curtain walls and less loss of coal. In other words, the perimeter of the pillar is less per ton of coal in the pillar.

MR. LUDLOW.—Do you pull the timber?

MR. GRADY.—The United States Coal & Coke Co. pulls the timber and the recovery is from 15 to 85 per cent., depending on conditions. But in general the timber is not recovered.

E. W. PARKER, Washington, D. C.—What was the average production per man of the United States Coal & Coke Co. as compared with the rest of the State?

MR. GRADY.—I am not in a position to give you those figures, Mr. Parker. I judge that the tons per miner per working day is about 250 per cent. of the average of the State.

CHARLES ENZIAN, Wilkes-Barre, Pa.—I am certain that Mr. Grady does not desire his statement concerning anthracite methods to convey the impression that such are the result of modern design. For the information of those who are not familiar with the conditions in the anthracite region, the present difficulties experienced in the recovery of pillar coal are an inheritance of the effect of old-time mining methods, rather than our own liking, or present mine design.

MR. GRADY.—It was not my idea to speak disrespectfully of the anthracite operators at all. I used the anthracite field as an illustration because of the large number of anthracite men present, and the wasteful practice with which we are all so familiar in the Schuylkill and Hazleton regions.

The paper by Mr. Whildin² at a recent meeting of the Institute shows very clearly that some anthracite operators realize very fully the enormity of the waste and the great benefit to be derived from concentration and having control of the robbing.

MR. RICE.—I would like to ask a question that has a bearing on the final recovery: that is, what percentage is permitted, by your company, in the advancing workings where the pillars are not immediately pulled? Also, what cognizance is taken of the depth of cover over the mine in figuring the percentage that may be taken out in advance workings?

In order to point the question, I refer to a matter relating to a very serious accident which occurred in Oklahoma a while ago, where so much had been taken out on advance workings in a steep pitching bed, that finally, when the cover became 750 ft. deep over the workings, the entire mine collapsed. I believe the Frick company of Pennsylvania has certain rules, but do not know whether they are invariably followed, but I understand it is its general practice not to take out more than 30 per cent. on advance workings. The company expects to take out the rest on the recovery of the pillars. Of course, the percentage would depend to a great extent on the depth of cover. The particular concern and interest is that it is a problem that arises from time to time in the West, particularly in mountainous districts.

² Steep Pitch Mining of Thick Coal Veins, *Bulletin No. 96*, December, 1914, pp. 2795 to 2819.

ARTHUR HOVEY STORRS, Scranton, Pa.—Is it not probable that the question, as to the proper amount of the seam to be taken out in the first mining, is one to be considered entirely independent of the thickness of the cover? The matter of so proportioning the area mined out to the pillars that the largest percentage of the seam may ultimately be won, at a minimum of total cost, is, I believe, of more importance than the fixing of the sizes for the pillars simply to carry safely the overburden until time for the final robbing. The size of pillars which will give the best ultimate return from the seam is, I believe, in all cases, much in excess of the size of pillars required for the mere support of the overburden until time for final robbing.

So far as I have known it, in the anthracite fields, the removal of from 30 to 35 per cent. of the seam in the first mining results in the ultimate recovery of the largest possible proportion of the pillars and at a minimum cost, and this regardless of the depth below the surface of the coal seam. This is particularly true where the workings stand for a number of years before robbing is commenced, for they will undoubtedly be found in better shape for the robbing work if the original mining has taken only a small proportion of the seam. Of course, if it is not expected that any robbing of pillars is to be done, then the matter of the calculation of the best sizes of pillars to be left for proper surface support is an essential one.

MR. RICE.—Actual safety is what I had in mind.

MR. GRADY.—I take it that what Mr. Storrs has in mind is this: The limiting factor in the proportioning of room to room pillar will be found not to be the thickness of the cover, but rather the economical use of labor and material, and it will usually be found that what is a proper extraction in this respect falls inside of the extraction allowable so far as superincumbent strata are concerned.

As a general proposition, I think Mr. Storrs is correct: *i.e.*, the limiting percentage of extraction on the advance workings is not determined by the amount of cover; but where the "nether roof" is very weak, or where it is very strong, 30 to 35 per cent. extraction in the advance workings precludes the possibilities of high recoveries unless they are immediately removed. Also where the depth of cover is very great the percentage of extraction in first workings must not be as much as 30 per cent.

MR. RICE.—One more point I might mention, an extreme case that came under my observation while doing some work for the Santa Fé R. R. some years ago. The Starkville mine of Colorado is working toward Fisher's Peak, the front point of a *mesa* south of Trinidad, the top of which is nearly 3,000 ft. above the coal bed. When the cover became over 1,500 ft. in depth a squeeze started, and it was found impossible to hold up the roof in rooms. They were obliged to drive to the boundary of the panels,

turning no rooms; then starting at the boundary drive up the rooms and pull them back quickly. The roof and floor are both strong sandstone; and even in the advance entries before rooms were turned, the coal would buckle out. That was an extreme condition, but I wanted to emphasize that with such a deep cover there should be some limiting percentage of coal area permitted to be taken out in advance with the expectation that a large percentage of the total coal would be obtained, chiefly on the retreat.

MR. TAYLOR.—In reply to Mr. Rice's question in regard to the practice, I will say that in the Connellsville district there is a peculiar condition existing in the coal, about halfway down a soft streak occurring in the seam. For instance, taking a section of coal seam about the middle, there is a space of about a foot which is soft where the strength of the coal is not more than half what it is above or below. The result is that you cannot leave small pillars; they have to leave immense pillars to avoid the crushing. That is not the same physical condition as in some districts where the coal has a practically uniform hardness from the top to the bottom. Consequently, in the Connellsville district rooms are driven on 80-ft. centers, and in the Pittsburgh district on 35-ft. centers, largely, and about 25 to 28 ft. taken in the advance and a pillar left for the retreat of probably not over three or four yards. At a little mine that I had charge of, where the covering was not over 150 ft., probably not over an average of 100 ft., we used longwall machines, driving the rooms 26 ft. to 28 ft. wide and leaving a small pillar about 8 ft. thick. We did the mining of the pillars with the longwall machines, making a cut along the rib of 35 to 40 ft. We would draw one of these pillars out in about a week's time. There was no necessity of using the longest length of cutter bar; the length we used undercut $6\frac{1}{2}$ ft. deep, and the coal would practically all come out without requiring the extra foot of undercutting on the gob side. When the coal was shot it would come down from the draw slate and leave only a small wedge-shaped piece of coal at the bottom in the back next the gob, and yet practically all of that coal was got out by machine mining. The coal was uniform in density from the draw slate to the bottom. There are several other districts, which are identical with the Connellsville district, where the soft binder, or soft streak in the coal, demands a very much increased pillar, not because of the overlying strata so much, as to prevent the crush by reason of requiring the rooms to remain a longer period than when operated continuously and uniformly and the coal mined out quickly. I mention that simply because you cannot be governed alone by the overlying strata of coal, but must take into consideration the uniformity of conditions.

Safeguarding the Use of Mining Machinery

Discussion of the paper of FRANK H. KNEELAND, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 97, January, 1915, pp. 61 to 65.

B. F. TILLSON, Franklin Furnace, N. J.—I think it may be of interest in respect to the information we have received in regard to the danger of machinery in mining operations, to state that in the particular type of metal mining which the New Jersey Zinc Co. is doing at Franklin, during the past year, although there have been other accidents, that method of mining was free from any accident due to machinery underground. It may give a different light as to the relative degree of the importance of machinery as varying with different types of mining.

ARTHUR WILLIAMS, New York, N. Y.—Mr. Kneeland's able presentation of this paper, it seems to me, bears out the suggestion which has been so frequently made that in this country we have a new profession, that of the safety expert or safety engineer, which calling, if properly carried out, would include also questions of sanitation. While Mr. Kneeland was speaking, it occurred to me that perhaps any one, however experienced he might be in mining generally, would pass through these mine shafts and these operating rooms, and the things which might cause accidents, which would cause accidents if the right conditions were to occur, would pass by entirely unnoticed by him, but when the safety expert goes into these places he sees these things which he immediately recognizes as being spots of danger and provides means for their elimination.

It seems to me the time is coming when the moral, if not the legal, responsibility resting upon any employer or controller or contractor of labor, for any accident resulting from the permitting of these danger spots to exist in his work, cannot be evaded. I say, at least, morally, but I think the time is coming when it will be made a criminal act for any employer or contractor to permit danger spots to exist where human beings are engaged in industry. I think the time is not far distant when we will look back with a certain degree of, I will not say horror, but a certain amount of wonder, why for so many years in all industries we have permitted these danger spots to exist. They are not confined to mining; they are to be found everywhere, in every industry. The conditions surrounding the flywheel on that high-speed engine, shown by Mr. Kneeland, will be duplicated, I think, something like 500 or 1,000 times in large buildings in this city. The conditions where men can put their hands in gearings and be caught in rotating machinery will probably be found to be duplicated in very many of the large buildings in this city and other cities. A single manufacturer by sending safety experts

through his works has reported that he was able in one year to eliminate something like 8,000 danger spots which were, at least, potential possibilities of serious accident to the human beings entrusted to his care.

Of course, we may think at first of the humanitarian side of this thing, but it goes beyond that. It seems to me there is an absolute definite responsibility resting upon any employer of labor to make the conditions surrounding the work conducted under his direction as safe as human experience can make it under the conditions existing at the time. But there is another side to it, and that is the safety, the higher efficiency of the working force, the better understanding between capital and labor, the advantage of leading the men to think and realize that the employers are not indifferent to their lives, health, and general welfare, and then there is the direct saving which comes from the elimination of accidents, in the larger amount of money that can be retained in the treasury of the corporation or employer. The Committee of Award for the Anthony N. Brady Memorial Medals, found and stated in their report that as a result of safety efforts in the case of one of the large electric street railway companies the expenditure for accidents, and on account of claims arising from accidents, had been reduced from 8.65 cents out of each dollar of gross income—8.65 per cent. of gross income—to 4.15 per cent.; in other words, the safety efforts of this corporation, with all that it meant in reduced accidents and better feeling toward the corporation, had actually resulted in saving about 4.5 per cent. annually upon the corporation's gross income, which might be more than 10 per cent., perhaps 10 to 15 per cent., annually saved upon the corporation's net income.

I believe that in the mining industry, or any other industry, where conditions are made safe, where pure air is supplied, where there is an abundance of good light, where machinery is safeguarded so that the men cannot slip into it, whatever their condition may be, the higher efficiency, added to the saving of the cost of accidents, may mean a large increase in the net returns upon the employer's operations.

Safety Methods and Organization of United States Coal & Coke Co.

Discussion of the paper of HOWARD N. EAVENSON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 413 to 430.

WILLIAM H. GRADY, Bluefield, W. Va.—Mr. Eavenson gave a comparison between accidents in the State of West Virginia and in the Pocahontas field. The figures quoted are, I presume, taken from the reports of the State Mine Inspector. There are no statistics available, in readily accessible form, giving a comparison between the Pocahontas field and the mines of the United States Coal & Coke Co. which are a part of the field.

It occurs to me that the paper would be more complete with these data and I will say that the mines of the Pocahontas field produced during the year 1913 about 15,000,000 tons, about 3,000,000 tons of which was produced by the United States Coal & Coke Co. This gives an indication of the extent of its operations.

The Pocahontas field lies largely in the counties of McDowell, Mercer, and Wyoming in West Virginia, and Tazewell in Virginia. Without consulting maps, I should say that all of the acreage under operation at present by this company lies wholly in McDowell County and that the average conditions of roof top and bottom in its mines are the average of the county, but that the worst conditions in some of these mines are the worst to be found in the county. The average conditions in Tazewell County are about the same as in McDowell County, except that the average thickness of the seam is much greater. The average conditions of top and roof and bottom are better in Mercer and Wyoming counties than in the mines of the United States Coal & Coke Co.; the worst conditions in these counties are not so bad as in McDowell and Tazewell counties and the best are better than those encountered in McDowell and Tazewell counties; also the thickness of seam is less in Mercer and Wyoming counties.

THOMAS W. DAWSON, Scottdale, Pa. (communication to the Secretary*).—The H. C. Frick Coke Co., the mines of which are in Fayette and Westmoreland counties, Pennsylvania, in the Connellsville coke region, has been pursuing a safety campaign for a number of years and a short *résumé* of its practices will be of interest in connection with Mr. Eavenson's paper.

Every official and foreman of the company is continually impressed with the fact that "safety" should be the first consideration, and all officials and their subordinates are brought together as one great committee on safety. Pamphlets showing the duties of the miner and the

* Received Jan. 9, 1915.

manner in which he may protect himself from danger and giving safety regulations for those working around machinery have been printed and generally distributed. Permanent danger signs are placed wherever there is the least possibility of an accident. When men are working in shafts, the "Men in Shaft" sign is placed so that no accident can be caused by mistake in moving cages. A similar sign is placed on hoisting engines and other machinery when it is being repaired. When workmen are cleaning or making repairs to the inside of a boiler, a "Man in the Boiler" sign is displayed outside and the steam valve for this boiler is locked and the key carried by one of the men until the work is completed. When coke-drawing machines are being repaired or cleaned, the "Do Not Move" sign is placed on the controller and the trolley wheel is locked and the key carried by one of the repair men until the work is finished. "No Clearance" signs are conspicuously displayed at all points about the plants where there is no clearance for a man between moving cars and obstructions of any character. Bridge guards and overhead warning signs are placed wherever needed.

In the mines, guide signs in various languages are posted at road junctions and on traveling ways, indicating the safest way out of the mine.

All machinery is safely guarded. These guards include locking devices for handwheels of valves, safety locks for electric switches, guards for water gauges; safety gaskets to be inserted in steam blow-off and feed-water connections when cleaning and repairing boilers; safety locking device for self-dumping cage; soap lubrication for air compressors; wagon guard and dumping platform for swing-gate mine cars; spooling device for tail ropes on haulages; stiles or protected crossings over rope and sheaves where necessary for men to pass; improved safety catch for cages; device for positively rectifying wagon catches on car hauls; self-closing hinges for shaft gates; steel galleries for runways over boilers, and safety platforms for operating electric larries. "Do Not Touch" signs are used about electric wires, indicating voltage of current; and "Do Not Pass Under" signs are used where there is danger in passing underneath structures. Steel doors are provided to drop over shafts which have wooden head frames or coal bins above them, should these wooden structures catch fire. The company has originated a device for automatically controlling high-pressure air compressors. When the temperature of the discharge air in the pipe reaches a predetermined point, showing that the pressure is excessively high, it acts on the thermometer and recording device, thus closing an electric circuit and energizing a solenoid. This moves over a tripping device, which opens the pilot valve, releasing the steam pressure on one side of the regulating piston. Thereby the valve on the steam feed pipe is automatically closed, shutting off the steam and stopping the compressor.

All hoisting engines are equipped with an automatic overwinding device, which acts directly on the engine, cutting off the steam and applying the brakes.

When it is necessary to clean the sump at the bottom of the shaft, the cages are hoisted to a clearance height and secured by iron pins, through holes in the guides; these pins are attached to the guides by chains, which prevent their removal when not in use.

At the surface landings of all shaft mines, a device is installed which prevents the gates from being opened when the cage is not in position at the landing. All hoisting compartments of shafts are lined at the cage ends. All cages and safety catches are periodically inspected, tested, and a written report made of the inspection. In no case is a hoisting rope kept in service longer than $2\frac{1}{2}$ years, even though apparently safe and in good condition. Frequent inspection of air shafts must be made to keep them open and free at all times from ice and other obstructions. A fireboss must make this examination and travel either up or down such shaft once each day, the mine foremen once each two weeks, and the superintendent once a month.

The company's rules require that in mines generating explosive gas not less than 500 cu. ft. of air per minute per person employed in the mine shall be provided at the intake and this must be so distributed that there will not be less than 300 cu. ft. per minute per person employed in each split at the working faces. No mine shall have at the intake less than 300 cu. ft. of air per minute per person employed, and at the working places at least 150 cu. ft. per minute per person employed. Measurements of air supplied are carefully made and reported to the general office once each week. Local officials at mines generating gas are required to keep air up to the working faces and to such other places where explosive gas might be encountered. At a number of the larger and more recent plants, the ventilating fan is operated by two engines, one on each end, and either of them powerful enough to operate the fan in case of failure of the other. All ventilating systems in the mines are ascensional.

The Clowes hydrogen-test lamp is used in all mines generating gas, for testing purposes. Samples of air are taken in gaseous mines and sent in copper cans to the company's laboratory, where they are analyzed. The results of the analyses are reported to the general office and to the mine. If these show a percentage of explosive gas which might have been detected by the Clowes lamp, the party making the test and reporting no gas is required to make an explanation.

Boreholes are frequently drilled from the surface to release any dangerous accumulations of explosive gas in the gob, where these cannot be removed by the mine ventilation. Shot firers have been employed to do all blasting by battery, and inspect all places where shots have been fired to see that there is no fire or other danger thereafter. Only the safest permissible explosives are used, and all tamping is done with clay.

All safety-lamp mines are examined on Sundays, holidays and lay-off days, and all mines which have been idle for more than two consecutive days are examined before operations are renewed. In the larger mines, wherever safety lamps are used, auxiliary escapeways are provided. In some instances these are stair shafts from the surface to the mine, placed in the active working sections, and used also for additional ventilation. In other cases, means of escape are provided by having connections between mines, which are closed by double iron doors. Frequent examinations are made to see that these doors are always in condition for use. Where coal dust occurs, a system of pipes and a supply of water under sufficient head and all necessary appliances are provided to dampen thoroughly the floor, sides, and roof of all parts of dry mines.

On rope haulage, a device is provided for disengaging the rope from the trip as soon as it is given slack. Brakes are provided for all mine cars and $2\frac{1}{2}$ ft. clearance is provided on all haulageways on one side; this side being indicated by a wide whitewashed strip on the rib.

Systematic timbering systems are devised and strictly followed. Printed regulations cover the system of timbering in rooms, headings, and rib and pillar drawing; these are worked out to suit conditions at the various mines. Timber is not set without caps or crossbars.

All mines have complete mine-telephone systems. Stables, pump rooms, haulage-engine rooms, shaft bottoms, underground offices and all such places where men might congregate are of fireproof construction and are kept clean and neat. No open lights are allowed in any building. Cans are provided for the reception of oily waste, grease, small quantities of oil, etc. All electric wiring is carefully inspected twice each year. All bare power lines underground and on the surface are properly guarded for their entire length by a neat wooden guard, so as to prevent the workman or his tools from coming in contact with the same. For the same reason, trolley wires for coke-drawing machines are placed at a sufficient height to make contact with tools unlikely. A system of checking men in and out of the workings is maintained at all of the mines. All abandoned places in the mines are fenced off.

The company employs four mine inspectors, one of them acting as chief. It is the duty of these men to visit each mine and thoroughly inspect it at least once every 60 days. When an accident occurs in or about any mine, the chief mine inspector promptly visits the scene of the accident, gathers all of the data he can relative thereto and makes a sketch of the surroundings. This sketch is put into permanent form, blueprinted and sent to each mine with a circular letter, giving a full account of the accident and making suggestions for the prevention of similar ones. This is discussed at the meeting of the local officials at each plant. Once each week, the superintendent of each plant and his subordinates meet and discuss mine conditions and operations in general

and especially matters pertaining to the safety of their employees. The discussions of these meetings are reported to the General Superintendent each week. General meetings are held at stated intervals at the general office, which are attended by the superintendent of each plant and heads of departments.

Projections for mine workings are made far in advance of the actual work, and the haulage and ventilating problems are planned so that when the mine is developed the best system is in use. Specifications are written for each mine, stating where and how the mining is to be done. The officials of the company make detailed inspections at intervals, insuring that their instructions and the best methods are actually followed.

A safety committee of three or four men is appointed at each mine, which inspects periodically the working places, roadways, ventilation and any other things which in its opinion might be the cause of an accident. The committee reports in writing to the superintendent of the mine, who forwards the same to the general office. These suggestions are immediately acted upon and all dangers reported, should there be any, are removed as quickly as possible. Three rescue and first-aid stations are maintained at the different plants of the company, which are fully equipped with the best apparatus and accessories obtainable. About 400 men have been thoroughly trained and qualified in both rescue and first-aid work, local contests being held by the different teams at various times.

Emergency hospitals, fully equipped, have been provided at a number of the largest mines and the company is contemplating the erection of one at each of its operating plants.

Tests are made frequently for gas above roof falls in gobs. Work is prohibited in any place in which gas is found, until after it has been removed. Mine inspectors instruct all new employees about dangers of their work. In the accompanying table is given the accident record of the H. C. Frick Coke Co. for the last five years.

Fatal-Accident Record of H. C. Frick Coke Co.

Year	Total Coal Produced, Tons	Fatal Accidents Inside of Mines	Fatal Accidents Outside of Coke Plants	Total	Based on Fatal Accidents Inside of Mines		
					Tons Mined per Accident	Accidents per Million Tons	Accidents per Thousand Employees
1910	16,567,609	30	10	40	552,253	1.81	3.20
1911	14,993,417	23	4	27	651,888	1.53	3.32
1912	18,596,502	33	7	40	563,530	1.77	2.91
1913	18,097,038	36	10	46	502,695	1.99	3.44
1914	11,725,448	12	4	16	977,121	1.02	1.46

C. W. GOODALE, Butte, Mont.—I had the pleasure of visiting Mr. Eavenson last October, when he showed me through the plants of the United States Coal & Coke Co., and explained the merit and demerit system by which such great results were obtained.

I have charge of the safety work of the Anaconda Copper Mining Co., at Anaconda, Mont., and was anxious to see if it would be possible to apply the merit and demerit system in the work of our company, but the trouble with us would be that under the conditions existing in our mines the foremen would not be able to take charge of the same number of men. In the mines of the United States Coal & Coke Co. it is possible to give each assistant foreman charge of about 25 men, but those who are engaged in the mining of copper will appreciate the difficulty in equalizing the number of men that the sub-foremen or assistant foremen can take charge of. In one case it might be easy for one assistant foreman to have charge of 40 men, working on one level, whereas in another mine in the same district it would be very difficult for a foreman to take charge of more than 20 men, the foreman having to go from one level to another. I have held the belief that there was a way of working out a proportional award, or something of that kind, based on the merit system, and depending on the number of men employed under each foreman, but we have not succeeded yet in doing that.

I thought Mr. Eavenson's work would show an increased efficiency, and judged from some of the figures I was able to work out that it would do so, but for the year 1913 it does not seem to have worked that way, although I presume his answer will be that they were not mining so much coal that year, and the tons of coal per man would be reduced on that account. I would like to have him explain whether the close attention to the accident record did not bring about increased efficiency.

I submitted a copy of his paper to one of my friends in Montana, who is engaged in coal mining. He did not care to be quoted, but in regard to the matter of standard timbering, he says:

"If I were to criticize the paper, it would be on the plan of standard timbering. While this is customary in quartz mines, I do not think it is applicable to coal mines where the variation is so marked between good roof and poor roof in the same mine, and my fear would be that in conforming to a standard plan some of the extraordinary pieces of bad roof might be passed over or considered as having been taken care of by timbers within a few inches of same and yet not just where they ought to be. This is partially provided for by the provision that extra timbers are to be used where necessary, yet a timber close by might in cases be considered as having taken care of the condition."

In regard to the question of the effect of intoxication, the Anaconda Copper Mining Co. is now publishing a monthly paper called *The Anode*, so named because all the copper shipped from the Anaconda mines goes

out in the form of anodes. In the February number is an item in which you may be interested. It appears as follows:

"Intoxication"

"Among the rules appearing on the bulletin boards at the mines is the following:

"Never go to work after drinking liquor, and if you must drink, Stay Home. Experience has proven that a great many accidents are caused from drinking intoxicating liquors."

(Some of you may know of the fact that on Sept. 1, 1914, the State militia was sent into Butte to stop some disorder, and Butte was put under martial law temporarily.)

"It will be remembered that from Sept. 1, 1914, to Sept. 14, all saloons in Butte were closed; that from Sept. 14 to Sept. 24 they were open only from 8 a.m. to 7 p.m., and that for the balance of the month they were open only from 7 a.m. to 10 p.m.

"The accident record of the Anaconda Copper Mining Co. shows the following significant figures:

"Number of accidents per 10,000 shifts:

July.....	6.22
August.....	11.25
September.....	4.21
October.....	7.58
November.....	6.07"

In other words, the number of accidents per 10,000 mine shifts dropped from 11.25 in August to 4.21 in September, and came up again in October to 7.58.

I presume it is the experience of everybody that the drinking of intoxicating liquors has a great deal to do with accidents. Men are not only liable to accidents themselves when drinking intoxicating liquors, but they are careless while under the influence of intoxicants, and this carelessness leads, perhaps, to the injury of their fellow-workmen.

I fully agree with Mr. Eavenson that education is a very important thing in the efforts to reduce the number of accidents. We found by a careful review of our accidents at Great Falls that 85 per cent. of them were due to the carelessness of the men themselves, or their fellow-workmen, and that not over 15 per cent., by close analysis, could have been prevented by any amount of safeguards or precautions introduced by the company. We are endeavoring to educate the men as far as possible to have them feel we are in earnest about the "safety first" movement. Recently we went through the mines and had a lot of photographs taken, also some moving-picture films, showing regular practices of the men, and special pictures showing bad practices. We intend to get

the men together and have them see these pictures as soon as possible. I realized the importance of this method of education at our October meeting of the Institute in Pittsburgh, when we were shown some very good pictures, taken by the United States Bureau of Mines, of accidents occasioned by bad practices in coal mines. I do not believe they have succeeded in getting the same kind of pictures in metal mines. It is important to have our men see some pictures illustrating bad practices in our own experience, and I hope to have these pictures on exhibition at an early date.

J. PARKE CHANNING, New York, N. Y.—I think this is the proper time to put on record a conversation which I had the other day with a prominent Russian banker who came to New York to represent his government in connection with certain financial negotiations. I was introduced to him by Chester W. Purington. What he said to me, which was so interesting, was in reference to the abolition of the use of vodka in Russia. He said that he was interested in some coal mines in Russia, and that when the war broke out 60 per cent. of their men were drafted into the army and the production of the coal mines went all to pieces. Simultaneously with that, the use of vodka was prohibited. The manager was in despair: there was a great demand for coal, and with his lessened force he did not know what he was going to do. In about a week the production of coal commenced to pick up, and pretty soon to increase very materially, and the banker telegraphed to the manager to know whether he had secured any more men. The reply came back that he had not secured any more men, but was still working with the 40 per cent. of his regular force. The production gradually increased, and to make a very short story of it, at the time that he left Petrograd, the mine was then working with about 50 per cent. of its normal force, about 10 per cent. having been added to the 40 per cent. which was left when the draft was made on the men in the mine. The production, however, was 130 per cent. of the best production at any time before the war.

He then told me the gradual steps. During the time of vodka drinking the men worked about four days a week. When they were deprived of vodka, they wanted something to do, and as times looked as if they were going to be hard, the men worked seven days a week. After they had worked for a month or so on that basis, they voluntarily came to the manager and asked that they might be relieved of the day's pay method and be put on contract. This he readily agreed to and put them on a tonnage basis. The tonnage still further increased, and then the men found that it was impossible for them to work seven days a week. You will remember after the French revolution they tried to cut out Sundays, and to work every day of the week. In the Russian mines referred to the men themselves finally concluded they would only work

six days a week, and so in their present condition this increased efficiency is achieved with six days' work a week. He said that the physical aspect of the men had marvelously improved, that their financial condition was better and they wore better clothes. He said the most remarkable thing was that some of the men 40 and 50 years old had started in to learn to read and write. So that outside of the question of safety, from the efficiency standpoint alone, you can see what the prohibition of alcohol produces.

The State of Arizona, I will admit much to my surprise, went dry the first of last January. On Jan. 17, in the Miami-Inspiration district, we had a strike in which the men were out eight days. It was the most orderly strike I ever heard of. Negotiations were going on during that time, and finally a sliding scale was arrived at, and eight days after the strike the men went to work. During the whole of that time there was not a single incident of disorder on the streets, during the day or at night. My manager wrote me that there was not even any loud talking. That is another example of what the abolition of alcohol will do.

B. F. TILLSON, Franklin Furnace, N. J.—Very successful results have been obtained by the New Jersey Zinc Co. at the Franklin mines due to the installation of a bonus system in the payment of the shift bosses or under foremen for the achievement they have made in the reduction of accidents. As a comparison, if we consider the first two months of the year 1913, no bonus system was then in vogue, although numerous safety measures of the general sort that most mines employ had been in use for a number of years. In May of that year a bonus system was instituted, whereby the shift boss under ground who had the best record for all the remainder of the year received a bonus of \$200. This stimulated the shift bosses to their best efforts. The rating was worked out rather elaborately on the basis of the employers' compensation laws, making a stated value, within limits, for various types of accidents, whether minor injuries or serious injuries, and a demerit representing that value placed against the shift boss's record; the shift boss having the least amount of demerits, for the year, received the bonus. It was very pleasant for all of us to know that the man who actually did receive the bonus was the man whose work had appeared throughout the year to show the greatest interest in safety precautions, although his territory was probably one of the most dangerous in the mines.

At the beginning of the year 1914, it was realized that there were some weaknesses in this system, inasmuch as the likelihood of the miner receiving a slight injury was as great as that of a serious injury; in other words, that good luck played an important part; and whether a fall of the ground merely bruised his foot or crushed his skull was often a question of good fortune. The system was reorganized at that time and

placed on a basis of the number of accidents entailing loss of time, measuring a loss-of-time accident as any which incapacitated a man from returning to work the day following that on which he was injured. Lost-time accidents were rated against the number of shifts of labor which were performed in any shift boss's gang during the month. The standard rating then adopted was that of 1.2 disabilities (accidents resulting in loss of time) per thousand shifts of labor which were worked in any boss's territory. This worked out very favorably for our safety work. As a comparison, taking the same basis of disability rating for the first four months, before any bonus system had been installed, approximately 25 per cent. of the shift bosses working in the mines would have received a bonus. During the eight months in which the lump-sum bonus was paid to the man having the best record for the year, the standing would have been that about 40 per cent. of those who could possibly have received bonuses would have received bonuses under that rate.

During the first nine months of the year 1914, 58 per cent. of the possible recipients of bonuses received them for having a disability rating of less than 1.2 per thousand shifts of labor worked. Of that 58 per cent. 36 per cent. of them had an absolutely clean slate, no disabilities in those months. In other words, 36 per cent. of the number of shift-boss months which were possible showed no disabilities in the working force from accident.

Mr. Goodale brings forth the feature of the application of this bonus system to metal mining, where we have territories at different levels, and the shift boss often has a large area to cover, and on different levels. The sizes of the gangs which were rated under this ruling average from about 30 men to as many as, in some cases, 75 men. Naturally, it seemed that the man who had the largest number of men under him incurred the greatest risk. The saving feature was the pro-rating according to the number of shifts worked, so that the man who had 75 men working in his gang could afford to have one or two disabilities per month and still be a participant in the bonus, whereas the man who had only 30 men working for him had to have a clean slate to participate in the bonus.

The question of the payments of bonuses to the bosses is one which has been more or less argued by those interested in safety work, and it seems to me it is of vital interest that it has worked out so well in this particular trial.

Mr. Goodale raises the question of the accidents based on rate of production, which seems to be the common method in coal mining, and more particularly in coal mining than in metal mining. It seems to me, although it may be of value in coal mining, it is of less value in metal mining, because of varied conditions of working in different parts of the mine, different methods employed in different stopes, and different methods of timbering employed under varied conditions of ground,

peculiar conditions of working in mines which have been developed, and have many old workings.

I am able to quote from some figures obtained at the Franklin mine during the past three years on the same basis, although it does not seem to me such figures are of as vital importance as those based on actual amount of time worked. In 1912, which was before either of these bonus systems was installed, the rate of fatalities was about 12.5 per million tons of ore and rock mined. During the year 1913, in eight months of which the lump-sum bonus system was installed, the rating per million tons was slightly less than 4, dropping from 12.5 to 4, and during 1913, in which the improved bonus system was in vogue, the rate of fatalities dropped to 1.3 per million tons of ore and rock mined.

Now, the fatalities, as previously noted, are often a question of good or poor luck. The system in vogue does not pay particular attention to fatalities, but takes into account any accident which is anything more than a very minor accident—any accident which requires a loss of time is of very great importance in the acquirement of a bonus. Based on the United States rating of a serious accident, one requiring more than 20 days of time lost, during 1912 there were 54.2 per million tons mined. During 1913 there were 26.6, and during 1914 there were 24.9, which shows that the serious accidents dropped to about one-half. The slight accidents, which required a loss of time of more than one day, and not over 20 days, dropped from the figure of 690 per million tons mined in 1912 to 255 in 1913, and 156 in 1914, which gives a good idea that all of the accidents have been greatly improved by this method of bonus payment, and I cannot emphasize too strongly my personal conviction that the bonus system of payment to shift bosses or under foremen is of great value in the reduction of accidents.

It is only natural that a man impressed with getting out a high tonnage, and seeing that conditions are generally safe, is apt not to take the time to caution a man about minor things, things which are perhaps occurring daily, because he feels that anybody with any common sense would know better than to do those things. He has probably told the man a dozen times already not to handle a bar or chute in that particular manner, and thinks if the workman continues to do it that way he deserves to be injured. But pay him a bonus, which amounts to 10 per cent. of his monthly wages, to avoid these accidents and it is giving him an opportunity to place this money in his pocket by being a kindergarten teacher and seeing that these men are instructed against the minor potentialities for accident.

WILLIAM H. GRADY.—In the discussion some doubt was expressed as to whether or not systematic timbering was superior to non-systematic timbering in the matter of accident prevention. The point of view

stated was that systematic timbering is designed to take care of ordinary conditions of top and that when extraordinary conditions of top are encountered the men may be less apt to guard against them, depending for safety on the systematic timbering.

My observations are that most accidents come from the ordinary conditions of top. For example, it is generally conceded that fewer accidents occur in the robbing than in the room work, although the top conditions are admittedly much worse.

The United States Coal & Coke Co., as I understand it, makes a practice of timbering systematically, even though the roof is good, to the extent the Advisory Board deems necessary to guard against ordinary conditions of bad top. The matter of guarding against extraordinary conditions of bad top is left to the judgment of the assistant mine foremen, checked up by the mine foremen and timber inspectors.

In non-systematic timbering the matter of taking care of all conditions of top is left to the judgment or opinion of men, some of whom are recent arrivals from Southern Europe who would not know a "kettle bottom" from a keg of powder. By systematic timbering ordinary conditions of bad top are provided for, and extraordinary cases of bad top are taken care of by an assistant mine foreman, who is employed because he has shown ability and proficiency in judging and acting in such matters. But that is a question of opinion as against opinion, and in the absence of facts we must depend upon opinions. However, when we have facts and statistics on a subject we should observe the facts and study the statistics. I have no doubt that a careful study of the facts and statistics, as presented by Mr. Eavenson, will lead to the conclusion that, in so far as the prevention of accidents is concerned, it does pay to timber systematically and exercise a high degree of supervision. From my perusal of Mr. Eavenson's paper, I am in doubt as to whether or not sufficient statistical data have been presented to form a conclusion in regard to the cost of systematic timbering. It is my understanding that the increase in the efficiency of the men and equipment, the more economical use of material, and the greater regularity with which the absence of slate falls permits the working of a room has more than offset the additional cost of timbering and supervision.

GEORGE S. RICE, Pittsburgh, Pa.—A previous speaker, after commending the plan of the U. S. Bureau of Mines in taking moving pictures, for educational purposes, of good and bad practices in coal mining, regretted that it had not carried on a similar campaign in metal mines. I am glad to state that the Bureau has during this past year taken moving pictures in many metal mines and quarries throughout the country. While the set has not been completed, the Bureau will ultimately have reels of many scenes quite as good as the first ones taken by the Bureau

photographers in co-operation with the United States Coal & Coke Co. in the Gary coal mines.

In regard to the question of systematic timbering as a measure of safety, I have been very much impressed with the advantage, as shown very strikingly in foreign practice, of systematic timbering; that is, fixed or regular spacing of props and other timbers, regardless how unnecessary it may appear to the miner or to the foreman. It seems to me that this feature represents one of the greatest differences between the safety practices followed in the coal mines in Europe and the coal mines in this country. I refer particularly to the good results obtained by close regular timbering in the mines of Pas-de-Calais district, France. Their record for small loss of life from falls of roof is very remarkable. As some of you know, there they timber without regard to the appearance of the roof, and have a most elaborate system of supporting the roof. It is expensive, but so far as the lessened number of accidents is concerned, it certainly pays, and I think it does also from a business standpoint. Roof and coal falls cause almost half the loss of life in American mines.

I was impressed by the features surrounding an accident, given in a report which recently came into my office from one of the Bureau engineers, in which a number of men had been killed by a large fall of roof. This fall had occurred on a main entry, and the report stated there had been no previous indication of any weakness in this roof as tested either by the sounding method or the more approved method of touching the roof to feel if there is any vibration while the roof is being smartly rapped with a bar. It indicated no weakness because there was such a large mass; but, when the fall occurred, it was found that there were slips parallel with the ribs of the entry or gangway which came up in a more or less inverted V-shape; the wedge-shaped mass had been supported by narrow lips resting on the edges of the ribs, on either side of the entry. The mass had thus been resting on insignificant supports, and when these weakened, it permitted the whole mass to fall on the men. That shows a case where, perhaps, systematic timbering, regardless of previous safe appearance of the roof, might have saved those men.

ARTHUR HOVEY STORRS, Scranton, Pa.—In regard to timbering, I think that not only should the timbering be done systematically, but use should be made of the foreign practice of "collars," or double timbering, instead of the single prop, which is responsible for much of the decrease in falls of roof. The caps or collars there used are very light, but large enough to catch the small slabs of roof which fall and cause so many of our minor accidents.

We have not in this country the autocratic power to stop the use of alcoholic drinks, which we have been told to-day has been used in other countries, but the efforts of one company in the anthracite field may be

of interest. The Delaware, Lackawanna & Western Coal Co. in a part of its field has made an appeal to the saloon keepers not to open in the morning until after seven o'clock, so that the men may go to their work without being tempted to drink liquor on the way. The request has met with considerable opposition from some of the saloon men and how it will work out finally I do not know. The coal people are hoping that the courts, in granting the next licenses to saloon keepers, will make it a rule, in connection with the license, that the saloons must be closed from midnight until seven o'clock in the morning.

The Lackawanna company also, some time ago, brought out what might be called a primer on accidents, in which photographs of proper and improper methods of doing certain work were shown. These were printed side by side and the things which should not be done were printed in red ink, as a sort of danger sign. The company also used this booklet as a primer to teach the foreigners the ordinary English, used about the mines, without attempting to make correct English sentences. That, I think, has been of considerable value.

I hope I may be pardoned for citing one instance in connection with a visit to Mr. Eavenson's mines, which I ran into, as showing that their officials do not simply make the rules, and let it stop at that. I went through the mines with the General Superintendent, Mr. O'Toole, and in one room a small fall of rock had knocked one prop out of place. In replacing it the assistant foreman had set it on top of a fallen slab of slate. The assistant foreman was not present at the moment, but meeting him two or three rooms farther on, he was at once given instructions by Mr. O'Toole to go back immediately and reset the prop properly, removing the fallen slate.

HARRY H. STOEK, Urbana, Ill.—Several of the speakers have referred to the use of lantern slides and moving pictures. The Committee on Junior Members and Affiliated Student Societies has been trying to get together during the past year a list of such slides and films. While the committee has obtained this information primarily for the use of the colleges and universities, it has occurred to me during the discussion this morning that they might be made of very much more general value. Mr. Goodale stated that his company had such a film. A number of mining companies run night schools for their employees, and while the larger mining companies can afford to get moving pictures, the small ones cannot afford it. If we can get a list of all available material of this kind and publish it in the *Bulletin*, and can arrange some exchange system by which the different companies could use this material, it might be of advantage, not only to us who are teachers, but possibly to the mining companies as well.

As to the practical value of these pictures, I would cite an instance

that occurred recently in connection with the miners and mechanics' institutes of Illinois. Films obtained from the U. S. Bureau of Mines were shown each night at a different mining camp in Illinois through an arrangement with the moving-picture places in the camps, by which these films were shown after the regular performance, or in one or two places as a part of the regular performance. After one of the performances a man came up and said: "I saw that same film some few months ago, and the next day, when we went into the mine, and I started to do something in a wrong way, my 'buddy,' who was a foreigner, said, "Don't do that, the pictures said not to.' "

LAWSON BLENKINSOPP, Landgraff, W. Va. (communication to the Secretary*).—During the two years I have been in this district as State Mine Inspector, I have taken much interest in the advancement of the safety-first method. Nowhere in the 11th district has such a move been made to discipline and organize for the benefit of the workmen as at the United States Coal & Coke Co.'s plants. Several points make this a difficult undertaking: First, they have very raw labor, and have to educate them; second, the place does not command men, as it is on a branch line isolated from the main line; third, men are prone to be antagonistic, at first, to safety methods, but they soon learn it is for their welfare, and become an interested party to the organization. The employee has a duty to perform and he is given to understand this, when he is employed. A book of rules and the mining law are given him in his language, and he is further instructed from time to time. The rules are enforced by the assistant mine foreman, who visits each working place three to four times daily and stays until his orders are complied with. Each employee is rated at a certain capacity, and should his efficiency fall below this standard, after a reasonable trial, he is dropped. This, in my opinion, is one of the surest methods to overcome the complaints of the miner, that he goes into the mine from day to day and cannot get cars to load his coal, that he cannot make a living, etc. At the United States Coal & Coke Co.'s plants it is the reverse; it is up to the miner to load each day the amount that is cut for him; each district assistant foreman knows how much coal he can get each day by 8 a.m. and that amount must be forthcoming; also each place must be cleaned up by quitting time, so coal can be cut again that night, unless a reasonable excuse is given.

I wish to make special note here as to roof conditions. In nearly all the mines roof conditions are dangerous. Of the 18 operations only five have good roof. The Pocahontas No. 4 seam has a rash following down from 3 to 5 ft. thick, and is dangerous in the extreme, requiring careful timbering. Systematic timbering is enforced in all conditions of roof, good and bad alike. Timber is removed from good roof places and

* Received Feb. 14, 1915.

used over again as the pillar is extracted. Many of the mines operating the Pocahontas No. 3 seam have a draw slate 30 in. thick. This is above the average thickness of slate in mines operating on the main line of the Norfolk & Western Railway. When roof conditions become dangerous mining is stopped, the place is timbered, or the loose roof taken down. They do not wait until the run is over, but do it at once. Nearly all operations have their own method of working. The latest economic method of the company is to extract advancing, and while the excavating is fresh pull all pillars, thereby eliminating the dangers due to crushed pillars, broken roof and brows of disintegrated draw slate. Many accidents are due to men going back too soon after shooting, without regard to the condition of the roof. This cannot be done at the mines of the United States Coal & Coke Co., as the assistant foreman does all the shooting in his district; after the shooting, the miner is not allowed to go back until his place is examined by the assistant foreman.

Ventilation is all that could be desired in a coal mine and the fixing of a safe minimum is an excellent regulation. It obligates the assistant foreman to keep all brattices and stoppings in his district in good condition, and should the company inspector find less than 12.00 cu. ft. at any last breakthrough the assistant foreman is called to account.

The premium system is good as it causes "competition safety;" also the posting of the accident list at each mine, showing minor, serious, and fatal accidents, is helpful. The official not interested in reducing accidents is not long-lived at this company's works. The weekly meetings of the officials are to discuss the prevention of accidents, by endeavoring to find the cause of any accident that may have occurred. A typewritten statement is posted on the bulletin board at face of each room, showing the cause of every accident that occurs and the method of preventing a similar one. The exhibition of moving pictures showing the various mining operations has been helpful in impressing on the miner's mind the dangers surrounding his work.

In justice to the other operators in my district, I would say that all have co-operated heartily in advancing the safety movement. Nearly all the mines have eliminated the dust problem; solid shooting has been practically eliminated. Clay is being used at most mines for tamping. Sanitary conditions have been improved, some operators providing wash houses for their men. Safety inspectors have been employed to aid in the prevention of accidents. In fact, many of the companies in the district are running a close second to the U. S. Coal & Coke Co. in the matter of safety.

HOWARD N. FAVENSON.—In reply to Mr. Goodale's question about the assistants and the average number of places attended by each, we do not have that uniform throughout all of our mines. The idea is to

make it as nearly uniform as possible, but the number of men whom each assistant oversees is regulated also by the distance which the assistant foreman must travel.

In addition to having a regular section in the mine to oversee, each assistant has in his charge a certain portion of the main-haulage roads and manways through which the men must travel to reach their working places, and he is responsible for any injuries occurring in these places. The assistant with a small number of men will have a larger section of haulage headings and manways than the one with a large number of men.

As to systematic timbering, we have a number of places, particularly in No. 3 seam, where portions of the draw slate over the coal are in the shape of truncated cones. Sometimes these "kettle bottoms" are only 2 or 3 ft. in diameter on the under side, and a foot or two on the upper side, and sometimes they run to 10 or 12 ft. in diameter, and it is almost impossible, when the place is first excavated, to discover them. The systematic timbering, in designing the layout of the rooms, is arranged so that each timber supports a cap piece, such as Mr. Storrs mentioned, which is likely to catch these kettle bottoms, and the posts are not over 5.5 ft. apart; at least, there is one every 5.5 ft. It is our standard to drive double-track rooms and right along each track we have a row of posts 3 ft. apart with the caps extending at right angles to the track, the theory being that they will protect the roadway to some extent, while in the remaining portions of the room the caps run parallel to the roadway itself. The timbers are kept not farther than 6 ft. from the face at all times; as soon as a portion of the coal is excavated the posts are set, and they are kept to the face at all times.

Replying to Mr. Goodale's other question about the efficiency having decreased, the figures of number of men employed, taken from the diagram in the table, include all outside men, and there were a great many of them on construction work in 1913, and the efficiency curve derived from those figures, as far as inside work is concerned, would not be strictly correct.

In 1911, by some time studies on the amount of time that was spent by the miners in loading, laying track, setting timbers for taking care of the roof, etc., and waiting on cars, we found that they were not working over 40 per cent. of the time. At that time our men were averaging about 2.5 to 3 cars apiece. By rearranging the haulage system and putting on a few more drivers, so that there would be no excuse for the miners waiting on cars, we were able, without increasing our piece rates at all, to increase the average earnings of the men \$1 a day. In other words, we increased the output from about 3 cars to about 4.5 cars per man per day, and for the year 1914, the whole year, our men averaged 16.4 tons

of coal per man per day at the working face. That does not include the drivers, or the machine men, but just the miners themselves.

Regarding the reduction in accidents shown, basing our figures on 1909, which was the first year we began this work, and taking the tonnage per injury of that year as a standard, I have the figures for the fatal accidents and serious accidents. We have reduced the fatal injuries 70.7 per cent., and the serious injuries 46.1 per cent. I have not the figures here for the minor injuries, but I think they would show at least a corresponding reduction. Last year we produced 458,900 net tons of coal per fatal injury inside the mine, and 367,100 tons per fatal injury, both inside and outside. In Mr. Dawson's discussion, he did not mention the fact that the H. C. Frick Coke Co. in the same year produced 977,000 tons of coal per fatality inside.

I have the figures for Great Britain, for 1913, the last available official figures taken from the inspectors' reports, and in that year they produced in Great Britain, including all the mines in England, Scotland, and Wales, 207,000 tons of coal per fatality inside. Taking all injuries, fatal, serious, and minor, and including everything from which a man lost over seven days' time, we mined 14,340 tons per injury in which any time over seven days was lost. I do not know how that compares with all of the foreign countries, but it is about ten times what the English figures show.

ARTHUR WILLIAMS, New York, N. Y.—Some figures presented to the American Museum of Safety a year ago showed that the average fatal accidents over a period of five years in European mining was one to about 168,000 tons of coal mined, and the average in this country for the same period was one fatal accident for about 186,000 tons of coal mined, taking the country as a whole, and including such States as Alabama and Oklahoma and British Columbia, in coal mining.

I understand that the German manufacturers have found that where they provide soft drinks for their men, in the absence of any municipal regulations, the tendency to drink alcohol disappears, or is greatly lessened, the men finding the soft drinks a good substitute.

G. S. RICE.—I think the movement is splendid, but we do not want to flatter ourselves too much by making the accident rating on the basis of tonnage, because there is a most important factor in the ease or difficulty with which we can get the coal. For example, in the longwall district of northern Illinois, the 1913 figures show that the daily output per employee is but 1.96 tons, whereas in Williamson County, in the southern part of the State, it is 4.06 tons per man. The miners are chiefly foreigners in both places; they interchange and belong to the same union; in other words, are just the same kind of men. The difference in the output is because in the northern part of the State the seam is thin, and

much rock must be hoisted, and otherwise a great deal of dead work must be performed in longwall mining, whereas in the southern part of the State the dead work is comparatively light, and the coal is largely cut by machine. Again, in Pennsylvania I see that in 1913 the soft-coal mine employees averaged 3.78 tons per working day, while the anthracite men averaged but 2.02 tons of 2,000 lb. per man. Would any anthracite operator or miner admit that the anthracite men were less competent, or the mines were less well equipped or administered than the bituminous mines of the State? I think not, and it is for similar reasons I believe it is a mistake to compare the number of individual accidents per thousand tons of coal produced in one district, State, or country with that of another district in which the conditions may differ widely. One might almost as well compare the accident rates in copper mines of the country as a whole on the basis of net tons of copper produced, which would be less a comparison of relative risk of life or limb than of percentage of copper in the ground.

In making comparison with European countries, both of accident rates on a tonnage basis and of costs of producing coal, we must take into consideration that in general the seams abroad are steep pitching and badly faulted; that machine mining can rarely be done; the work is deep and the gaseous conditions more serious than in this country except in our anthracite district. The seams are thin, and even when thick the advantage of cheap mining is lost to a great extent because the operators are required to pack or stow the excavations either by hand or hydraulically, to prevent excessive subsidence of the surface; furthermore, the mining of the thin seams, which may be done at monetary loss because of the dead work, is carried on simultaneously from the same shafts as the thicker seams, which decreases the average yield per man, and increases the average cost of the output.

While the method of comparing annual individual accident rates on the basis of 1,000 employed is open to some objections, it better enables one to consider the relative risk of the different kinds of employment in and about any one mine, and also of different mines, with the same or with other conditions. To better compare the actual risks, it seems advisable to standardize the relative exposure on the basis of time; that is, if the men at one mine, on an average work 200 days of 10 hr. in a year, and in another mine 300 days of 8 hr., the men of the first mine would be exposed for 2,000 hr. during the year, and in the second mine for 2,400 hr.; the men in the second mine would have had additional exposure in the ratio of 24 to 20. A standard year of 2,400 hr. might be selected and the rates figured to this basis. All mines keep account of the number of hours run per day, so that there would be no difficulty in reporting the hours per year, and figuring accident rates on that basis.

CHARLES W. GOODALE.—In reply to Mr. Rice, I will say that we have adopted in the mines of Montana the basis of so many accidents per 10,000 shifts, for the reason that every shift represents a risk, which may be incurred either in development work which has to be done, or in the breaking of ore.

Reviewing what we have done in the "safety first" work, and comparing 1914 with 1913, I will say that we have reduced the fatal accidents 35 per cent. on the basis of 10,000 shifts, and the serious accidents 26 per cent. In the case of minor accidents, we have not been so successful—we have only reduced them about 3 per cent. We have adopted the plan of getting the shift bosses together every month and making them stand up and tell how each accident under their jurisdiction occurred, and every one present is then called upon to make suggestions as to how that kind of an accident may be avoided in the future.

HOWARD N. EAVENSON (communication to the Secretary*).—From the tenor of the discussion of this paper, as well as remarks made to the writer by various members since the meeting, the general opinion seems to favor the system of giving bonuses to foremen for work of this character and, apparently, the general opinion is that all men engaged in the handling of labor should be in favor of a system of this kind. This, however, is not the case, as some take the view that where a foreman is paid to do a certain thing it should not be necessary to offer him additional inducements in the way of bonuses for performing what is apparently a portion of his work. This may be true if the foreman is receiving a fair compensation for his labor, but the answer to this point of view is that the attitude of the public within the last six or seven years has entirely changed with reference to industrial injuries. Under the old theory the workman was supposed to assume all risks of the employment and could only recover damages for injuries received when negligence on the part of the employer could be shown, and under this theory safety measures that were taken were adopted purely from a humanitarian point of view. The new theory, now almost universally held, is that the business in which any workman is engaged should be responsible for the cost of all injuries received and that the cost of all such injuries is a legitimate charge against the cost of production; and that compensation should be made for all injuries, no matter how occurring. In accordance with this attitude, employers must use all known safety devices so there will be fewer injuries to the workmen and thus necessarily lower the cost of production. In addition to the humanitarian side, the employer now finds it a commercial proposition to keep his works or mines as safe as they can possibly be, not only for the sake of the money saved in the compensation claims,

* Received Mar. 18, 1915.

but also from the increased efficiency and better results achieved in all lines from the men when working under safe conditions.

In all lines of endeavor this same change in attitude can be seen. The old theory was that the employee must follow instructions or be made to suffer for not doing so by being reduced to an inferior position, suspended for a definite period, or discharged. The new theory differs from this in the fact that ordinary services receive the regular rate of compensation, services above the ordinary an extra rate, and the failure to perform ordinary services is treated as rather due to incompetency or inability than to neglect, and in any of our large establishments it is customary to find a workman changed from one position to another until he at last finds work he can do satisfactorily, or is finally discharged from the rolls.

All energetic management to-day is trying not only to educate the workmen in safety measures but in efficiency, sobriety, and general industry.

Enlarging the Worth of the Worker and the Perspective of the Employer

Discussion of the paper of J. PARKE CHANNING, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 99, March, 1915, pp. 529 to 538.

FRED H. RINDGE, JR., * New York, N. Y.—It is interesting to note that although this movement was started only seven years ago at Yale, it has spread in that brief time to 200 colleges and technical schools, and thousands of men are interested in it. There are 3,000 graduates who became acquainted with the movement in college days and are now promoting the industrial betterment idea all over the country. Some of these men are employed entirely in this work, and the number of such workers is constantly increasing.

We feel we are really doing a very fundamental thing in getting so many coming engineers and prospective employers of labor to interest themselves in these things which we have been discussing this morning; these things which are so vital to efficiency and success in any line of business or engineering to-day.

Of course the institutions located in our large industrial centers have the greatest opportunity for practical service. But I have just returned from one college, Pennsylvania State College, where 300 engineering students in three days signed up as interested in this movement and as anxious to take active part in the work this summer. That is one of a number of colleges situated in towns where there are no industrial opportunities. Therefore, their field of service must be where they work

* Traveling Secretary, Industrial Service Movement.

or live in the summer months or after graduation, and a large portion of those 300 men will be active this summer. There are men here who, if time permitted, could speak with conviction of the influence of this movement on their respective colleges.

There is another thing that should be especially noted, and that is indicated in Mr. Channing's paper, on the fifth page, where there is a rather specific illustration of the reaction on the student, the coming engineer, showing in just what way the experience helps him, and then the paragraph toward the close of the seventh page, and from there on through to the middle of the ninth page, further illustrates the other activities which the movement is promoting.

Mr. Channing did not take time to mention a number of the other practical features involved in this work. Possibly one of the most significant is that at many of the colleges and technical schools there is given a series of special lectures on the human side of engineering, accident prevention, and other things, which these coming employers must be interested in. One significant fact is that a number of colleges are planning to put in the curriculum required courses on the human side of engineering. That is a very important step and one which will be especially welcomed by practical business men and the alumni who now feel so greatly the lack of such instruction in their own training. When we have in our engineering schools a course on the humanics as well as the mechanics, it will be a big advantage to industry. It is surprising how the already full curriculums of the colleges can be readapted to include such important instruction as has been indicated.

It is clear that the movement is striking at some fundamental problems of the day and is helping constructively in many ways. Additional information regarding the whole plan may be obtained from the publications of the Industrial Department, 124 East 28th St., New York, which will be sent on request.

S. A. TAYLOR, Pittsburgh, Pa.—A little experience we are having in Pittsburgh may be of interest. About three years ago I took up with the representatives of the School of Mines of the University of Pittsburgh, the miners, and the Y. M. C. A., the question of working out a system of co-operation for the different mining camps around Pittsburgh. The result of that effort is interesting. We first tried to establish in the school a Saturday class for the men from the mines, in order to prepare them for the examinations they would have to pass for mine foremen or fire boss. The result was not a complete success. We found the men would not go into the school on Saturday, although a number of their employers were willing to pay their way and give them their day's wages. We then followed the saying of Mohammed, that if the mountain would not come to us, we would go to the mountain, and the result of it was that

we established a series of night classes at the different mines in connection with the Y. M. C. A. work. The result was very satisfactory. During the year 1912-1913 we had 137 students in these night classes—I cannot give you the number of foreigners in those night classes, but there were a number. Some of the men had also previously taken some work in connection with the Y. M. C. A. classes. At the expiration of the first year 50 men out of the class of 137 tried the State examinations for mine foremen and bosses, and all but two passed.

The plan worked this way: The Y. M. C. A., through its organization, would arrange for the class, and the university would furnish the teachers. We had only two men we could spare for this work, and inasmuch as they would be obliged to move from place to place every day in order to take care of the night classes, we arranged with each of the classes to bear the expense of the instructor, simply his car fare and his hotel bill. The university paid the salary of the instructor.

A small, inexpensive hand lamp helped to make the system of teaching effective. By means of the reflector the illustrations, taken from advertisements in various technical journals, were thrown on the screen, and the machines and their operation explained. The instructor could carry this lamp with him from place to place. The result of that trial was a demand for this form of instruction so great that it was impossible for the university to carry it on, the funds being insufficient. I think the University of Illinois and the University of West Virginia are carrying on similar work.

There is another feature of this work which is worthy of mention. The manager of one of the companies to whom I proposed this matter said: "Yes, I will be glad to co-operate in this work, because I have carried out similar work in connection with the Y. M. C. A." He told me that by getting his men interested in this class of work he had reduced a great number of the little agitations and complaints which came to him through the labor union. Men who would ordinarily go to the labor union meetings and talk and agitate matters, instead went to these night schools, and the result was that his troubles at the mine were greatly reduced.

In connection with that work we have established at the university a co-operative system, which all students taking the engineering course must enter to the extent, or the equivalent, of 10 weeks. There are only two weeks' vacation in the entire year. One-quarter of the entire year must be spent in the shop, in the engineering course, and in the case of the mining students, in or around some of the mines. We have found this an efficient way of bringing the students in close contact with the men in the various occupations. I believe it would be worthy of consideration by any of those who are so situated that they can avail themselves of such an opportunity.

H. H. STOEK, Urbana, Ill.—I think it is only right I should say a word, because Illinois was the first State in which such work was authorized to be done by the State, or through a State organization. Several years ago authorization was given to carry on the miners' and mechanics' institutes, but no appropriation was made, and only 12 months ago an appropriation of \$15,000 became available for carrying on the work. Since that time 17 night schools have been established in Illinois, and about a thousand men are now receiving instruction in these schools. The work has been fully described in the *Bulletins* of the Miners' and Mechanics' Institute. There is a decided demand for such work and we find in Illinois that its scope and extent are limited simply by the amount of money available. We have asked the Legislature to give us \$52,000 next year to carry on the work, and if that amount is granted, we can put a night school in every important mining center in the State of Illinois. The difference between what we are doing now and what can be done depends on the appropriation. We have had a number of inquiries from mining companies wanting young men trained for night-school and similar extension work.

ELMER L. CORTHELL,* New York, N. Y. (communication to the Secretary†).—This paper has interested me greatly and I fully believe in the excellent work outlined, and I may say I am in a small way assisting in it.

I know of the good work being done along this and other lines by the Y. M. C. A., particularly in other countries. I was one of the founders of the branch association in Buenos Aires in 1901, where now there is a membership of over a thousand and where the branch owns a very fine building for its various purposes. Again, I was in Switzerland at the time Dr. John Mott undertook to form several branches of the International Christian Students' Union. I assisted financially in establishing a branch at Berne, and I am still one of its annual supporters.

The reflex action upon the students themselves in the Industrial Service Movement which Mr. Channing describes is a very great benefit to them. It could not be otherwise, for it leads them to be useful men and to get into the habit of *service and self sacrifice*, and this lays a foundation for not simply *making a living*, but for *living a life*.

I think any engineering organization like yours could properly support such a movement and appoint a committee on "Industrial Service."

PHILIP W. HENRY, New York, N. Y. (communication to the Secretary‡).—Mr. Channing's paper is of great interest and should be read carefully by all those occupying executive positions and by those who ex-

* Civil and Consulting Engineer. † Received Mar. 22, 1915.

‡ Received Mar. 31, 1915.

pect to fill them, because it shows what an important part the knowledge of human nature plays in dealing with labor and kindred problems. Such knowledge cannot well be gained from books, but must be gathered at first hand, by direct personal contact. Fortunate is the young engineer who—perhaps by force of circumstance—begins his career in a position where he associates on equal terms with the laboring men—using this term in its broadest sense. The insight which he thus gains into the real feelings and thoughts of the men who compose the membership of labor unions will be of great value to him later on when, by virtue of his technical education or superior natural talent and industry, he is occupying a position of responsibility, where his success may depend largely upon his ability to secure the hearty co-operation of his employees.

As most engineers are denied this intimate personal contact with the laboring man, the opportunity afforded by the Industrial Movement, as outlined in Mr. Channing's paper, should be heartily welcomed by the student engineer; for although primarily organized in behalf of the laborer, with the student as instructor, the teacher, rather than the pupil, is the one who obtains the greater benefit.

As engineers are coming more and more to fill executive positions, the knowledge of human nature becomes more and more important. Unless an engineer is able to select and place the proper man for the positions under his charge—a round man for a round position, and a square man for a square position—he will never hold for any length of time an important executive office. In other words, he must be an engineer of men as well as an engineer of materials. It is just as serious a mistake to select for mine manager a man who has only the qualifications for salesmanship—and *vice versa*—as it would be to select iron wire instead of copper for a transmission line.

In the past little emphasis has been placed upon the importance of the human element in engineering operations, and the Institute is to be congratulated on having a paper on this subject presented by a man of such broad experience and high standing as Mr. Channing.

High Blast Heats in Mesaba Practice

Discussion of the paper of WALTHER MATHESIUS, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 99, March, 1915, pp. 539 to 555.

JOSEPH W. RICHARDS, So. Bethlehem, Pa.—This paper answers partly the difficult questions which have come up as to why it is not economical to use higher blast temperatures in the smelting of Mesaba ores, such as the high temperatures used in European practices, and the answer is very plain that the ores will not stand the higher temperature. But the paper does not go into detail as to what special difficulties were found with the higher temperature of blast, and I would ask the author to kindly specify them.

W. A. FORBES, New York, N. Y.—A large part of the troubles that furnace men experienced in using high-blast heats in smelting Mesaba ores in the early days of using Mesaba ores, was due to the attempt to use these soft ores on furnaces having the same lines as the furnaces used in smelting the hard Old Range ores. As the result of observation and study, the dimensions of the lower part of the blast furnaces have been changed and the difficulties of the blast-furnace men in operating with a large percentage of Mesaba ore and high-blast temperatures have been largely overcome. The changes I refer to in particular are: (1) steepening the angle of the bosh; (2), decreasing the height of the bosh; (3) increasing the diameter of the hearth.

JOSEPH W. RICHARDS.—Mr. Forbes's answer is quite satisfactory, and the moral which it points is that when the furnace is run with a higher temperature in the smelting zone, the furnace lines which were adapted for the lower temperatures are not necessarily those best suited for the higher hearth temperatures. If enriched blast were used in a furnace, and higher temperatures run, it would be found necessary to further modify the lines of the furnace to meet the new conditions and to get the maximum output with economy.

J. E. JOHNSON, JR., New York, N. Y.—It seems to me this is one of the few papers which could well have been a little longer. The data which Mr. Mathesius has given us, I think should be considered in connection with the paper which Mr. Brassert read before the American Iron and Steel Institute last spring, in which he described how they had obtained remarkable results with furnaces with steep but very short boshes; the angles have been raised to 80°, and the boshes made so short that they no longer appear as a conspicuous part of the shape of the furnace.

The use of high heats involves an increase in the temperature of the hearth, relative to that of the bosh, and that to a certain extent might

be expected, because the ratio of expansion of the gas from its temperature at admission to the temperature at which it leaves the bosh is less, of course, in proportion as the temperature at which it enters is higher; that is to say, the greater its volume the greater its temperature at entering, but it is the same at leaving the bosh, because it leaves it at the same temperature in either event.

There are probably other considerations in regard to the characteristics of the shape, which we do not understand, but which have been worked out at South Chicago, to enable them to get these results.

Another factor which bears upon the case, is that they have a remarkable coke—the making of coke has been reduced to a science in recent years; they can make a coke of just the characteristics required, neither too hard nor too soft, too porous nor too dense, to suit their conditions, and by that means they cut down the solution loss, and are enabled to make other improvements which they could not use alone. This is one of the cases where by making three or four improvements you can make another operative, which would not be permissible without them.

One feature in the heat balance which Mr. Mathesius gives is that he uses a very high value for the total heat of the slag and the iron, much higher than is quoted in any of the authorities. I have written to ask Mr. Mathesius if he is positive about the correctness of these figures. I hope he is, because if so it will explain a great many things which have not been explained up to now. For instance, why do we take such a large increase in coke for a comparatively small increase in slag volume? It has not been explicable on the basis of the low value of heat of slag used in the past, but is entirely explicable if his figures are right. Some experiments I made in a crude way some years ago seem to indicate his figures are correct, and I hope he will be able to assure us that they have been checked out and are correct; if so it will be of very great value.

Another point I notice is that the solution loss of the coke, figured from the gas analysis, which is undoubtedly correct, is about 25 per cent. of all the carbon. Twenty-five per cent. of the carbon never gets down to the tuyères, and this results in a very great loss of the heat that we could get in the hearth and also of the heat which we could get in the shaft. The effect of solution loss is complicated. I have been making some investigations along that line recently, and find that some of the old ideas are not in accordance with the facts. For instance, it used to be thought that a low top temperature was the sign of a good working furnace; many people put water in the top of a furnace to hold the dust down and control the top temperature with it. The fact is, on the basis of some figures which I think are absolutely incontestable, it can be shown that the less solution you can have under any conditions within reason, the higher the top temperature will be. That coincides with our experience

with Old Range ores, when the solution loss was smaller than it is now, that the top temperatures ranged higher; in other words, the effect of the solution of carbon by the oxygen from the ore is a very decidedly cooling reaction; it cools off the top gases of the furnace very rapidly.

Such data as Mr. Mathesius has published here are very rare and almost beyond price when one is attempting to find out what really goes on inside the blast furnace. I think the operators of the South Chicago Works are to be congratulated in the economy effected, and the paper is deserving of the greatest of praise because it gives detailed information which is literally almost priceless.

JOSEPH W. RICHARDS.—Referring to what Mr. Johnson speaks of as the abnormally high amount of heat in the iron and in the slag, taking them as 510 B.t.u. per lb. and 900 B.t.u. per lb., my impression is that those are about average values and are not particularly high. They amount to 285 and 500 calories per kilogram, respectively, which are not abnormal.

I am pleased that Mr. Johnson called attention to the high solution loss; in other words, 25 per cent. of the carbon does not get down to the tuyères under these conditions, which is contrary to what were supposed to be conditions of economical working in the olden days. The real point is, that if the furnace works hot enough without burning all the carbon down there, why do you want to burn it? If you can use some of it above in doing direct reduction, and at the same time get all the heat you need down below, you are better off, and you are running the furnace with so much less carbon per ton of pig iron. There is no use in burning all the carbon down at the tuyères, if you are getting heat enough in the hearth without burning it all. I think the high solution loss is exactly an index of the economy of fuel in this furnace.

W. A. FORBES.—Answering Mr. Johnson, I would point out it is not changes in the diameter of the bosh which have resulted in improved conditions; many of our furnaces have the same diameter of bosh on the Mesaba ores as was used on the Old Range ores; it is the changes in the bosh angle, the height of the bosh, and the diameter of the hearth which have been beneficial.

In regard to coke, it is quite true that, due to the introduction of the by-product coke oven, an improved grade of coke is used at our blast furnaces in the Chicago district. At the same time, the two furnaces treated in this paper operate with the same coke as the other blast furnaces at the same plant. It is at those furnaces which have had their lines modified where the best results are obtained with high blast temperatures with the Mesaba mixtures.

In regard to the top temperature, the reason that water, in our practice, is added to the stock entering the top of the furnace, is primarily to reduce the amount of dust that is blown over with the gases.

H. P. HOWLAND,* South Chicago, Ill. (communication to the Secretary†).—The paper is well worthy of our careful study, not only because of the interesting data in Table I, but more particularly because it brings up the matter of correct stove design.

Table I shows that at South Works they have been able to operate on remarkably low coke and high heats. The question of low coke operation discussed from the cost standpoint is a large one. Where the furnace under discussion is so located that the surplus gas has a large value, the saving in lowering the coke, effected by raising the heat, is one that needs careful consideration. The only way to get the full benefit of this saving is to bring the gas-consuming apparatus to the very highest efficiency.

Mr. Mathesius rightly, therefore, follows his table showing their remarkably low coke consumption by a discussion of stove design. To reap the full benefit of the former, the latter must be much different from that of to-day.

Let us assume two furnaces using washed gas and stoves of 50 per cent. efficiency:

Furnace A: Coke consumption, 1,700 lb.; blast heat, 1,200°; gas, 82 B.t.u.

Furnace B: Coke consumption, 2,100 lb.; blast heat, 800°; gas, 92 B.t.u.

Neglecting the saving made because of the lower amount of blast needed by the low-coke furnace, the heat value of the gas per ton of iron is distributed as follows:

Total B.t.u. in gas A	= 9,922,000
Total B.t.u. in gas B	= 13,800,000
B.t.u. consumed to heat blast: A	= 3,534,000
B	= 2,680,000
Percentage of gas used on stoves: A	= 35.7
B	= 19.4
Surplus B.t.u. for power: A	= 6,388,000
B	= 11,120,000

Power lost by low coke on A: 4,732,000 B.t.u.

Saving by low coke on A is, therefore, 400 lb. of coke at \$5 per gross ton = 89c.

Loss by low coke: Assume 1 lb. coal = 11,000 B.t.u. 4,732,000 B.t.u. = 430 lb. coal, at \$2 net ton = 43c.

Actual saving of low coke practice: 89 - 43 = 46c.

These figures, of course, only apply to a district where the coke and coal prices are about as above.

There are probably many incidental savings aside from the above that can be attributed to the low coke consumption, such as greater daily tonnage, greater tonnage on lining, better furnace practice in general and a more uniformly working furnace.

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† Received Feb. 13, 1915.

However, to gain the full benefit of our low coke consumption, the 43c. loss, due to higher coal consumption, must be eliminated. This can only be done by making the gas from Furnace A do the same amount of work as that from Furnace B. In other words, we must increase the efficiency of our gas-consuming apparatus.

If we increase our stove efficiency from 50 to 75 per cent. we will use 2,356,000 B.t.u., or 23.7 per cent. of the gas. This leaves for power 7,566,000 B.t.u., which must do the work done on the B Furnace by 11,200,000 B.t.u. We, therefore, must increase the efficiency of our power-generating apparatus 50 per cent., as, for example, from 8 per cent. to 12 per cent.

From our experience in developing the efficiency of our old stove equipment at this plant, and from what we are continually learning on the topic, it does not seem to me the day is far off when stoves will be so built as to give us an efficiency of 75 per cent. and give it continuously. There is unquestionably a chance for much increased efficiency along the line of power-generating apparatus. It seems to me, therefore, that we blast-furnace superintendents are in a position to say that if the day approaches of high heats and low coke consumption, we will be able to turn the whole of the saving into dividends. There will then be the added advantage that if our furnaces for any cause should again have higher coke consumption, under the increased stove efficiency, the loss will not be as great.

From my own observation, I would say that there is less reliable knowledge and data about the use of blast-furnace gas than upon any other topic of equal importance in the steel business. Few blast-furnace men know what their stoves are doing, aside from the fact of the heats they are getting. This is due to the fact that when on high coke consumption our gas has been so high in heating value and of such quantity as to go far toward meeting the plant demands. As long as this condition obtained, it was perhaps natural that not much improvement would be shown. Such was the condition at this plant. When, however, we began to operate on our present basis, namely, one furnace instead of three, and the coke consumption on this one in the neighborhood of 1,700 lb., the coal smoke from our boiler house, as well as the monthly steam cost, was a very forcible reminder that not all of the coke saving was net saving.

The natural result was an immediate attempt to improve the efficiency of our stoves. We were fortunate in having a gas-washing plant, the design of which had been changed while relining the furnace, enabling us to wash the gas for the stoves and use unwashed gas on the boilers. As Mr. Mathesius points out, washed gas makes possible the design of a stove from a heat-economy standpoint and not, as was heretofore compulsory because of dirty gas, from the standpoint of easy cleaning.

To me this seems its greatest value, though nearly as great value I would place upon the fact that with the washed gas we are able to measure its volumes accurately, and know much more exactly what we are accomplishing. It is almost impossible to get accurate data on the use of dirty gas.

For the last three months we have been running some tests upon the efficiencies of the stoves. We have made about 50 tests that we consider accurate, these covering several different conditions, as, for instance, five-stove operation, four-stove operation, and attempted three-stove operation. These tests cover operation on atmosphere burners and burners using forced air.

We feel well repaid, both in knowledge and money, for all we have done. When we started the tests there were five stoves in operation: four central-combustion and one side-combustion chamber. The former were small inadequate stoves; the latter a new one with 9-in. checkers. The central-combustion stoves gave an efficiency of 40 per cent. and the other 60 per cent.

By the use of checkers in the combustion chambers the efficiency of two of the small stoves has been increased to 50 and 57 per cent., respectively. We have now eliminated one small stove and are using four stoves, having a combined heating surface of about 130,000 sq. ft. On the two small stoves of 50 and 57 per cent. efficiency, we are using pressure burners. We hope by installing this burner upon the stove of 60 per cent. efficiency to use only three stoves. If this is accomplished, and we are confident it can be, the gas will be burned in a stove plant consisting of three stoves of 50, 57 and 60 per cent., or a combined efficiency of 56 per cent. We were previously using four stoves of 40 per cent. and one of 60 per cent., or a combined efficiency of 44 per cent. We will, therefore, increase our stove efficiency 27 per cent., and consequently use 27 per cent. less gas on our stoves. During the period of these stove tests, covering the last four months, October, November, December, 1914, and January, 1915, the practice on this furnace closely approximates the following figure:

Daily tonnage.....	564
Pounds of coke per ton.....	1,700
Cubic feet of air per minute at 62° F. =	36,000
Blast heat.....	1,100° F.
B.t.u. in washed gas.....	82

It seems to me that the furnace superintendent with a low-coke furnace must work along such lines as these. It should not be a question so much of tearing down old stoves and building new ones, as an attempt to increase the efficiency of the present equipment, and thereby develop the proper design of the stove of the future. The problem is a very different one from that of simply erecting a new stove plant of larger

capacity of the old design, for we must secure not only the higher blast heat, but higher stove efficiency.

In our use of the pressure burner, we have found that it, in itself, does not necessarily add to the efficiency of the stove, and, consequently, does not necessarily result in any gas saving. The question of gas saving is, and always will be, decided by the efficiency of the stoves, as regards ability to absorb and give up heat. If it is then desired to reduce the number of stoves, the pressure burner will be most essential.

This burner is, however, of great value for two principal reasons:

(1) Accurate mixing of air and gas, with consequent complete combustion.

(2) The large amount of gas that can be forced into a stove. This means that if we have stoves of high enough efficiency we can run on two stoves.

When designing a new stove plant on the basis of washed gas, the following points are most essential: (1) Small checkers of proper design. (2) Large ratio of heating surface to shell radiating surface. (3) A well-insulated shell. (4) A burner designed to force a large amount of air and gas into a stove and mix it correctly.

There is no reason why stoves with 75,000 sq. ft. of heating surface cannot be designed. This stove properly insulated should give an efficiency of 75 per cent.; 11,000 cu. ft. of 82 B.t.u. gas, burned in one such stove per minute for 60 min., will heat 36,000 cu. ft. of air to 1,125° F. for 60 min. The question of high heats will then be simply a question of forcing more gas into the stove. The gas consumption on the above basis is only about 23 per cent. of the gas produced by a 550-ton furnace running on 1,700 lb. of coke.

We have been able to burn 9,000 cu. ft. of gas per minute in one of our small stoves, on a rather temporary pressure burner equipment. This is easily twice what this stove used with the old type of burner.

The question of low coke consumption and high heats is going to bring to our attention very forcibly this matter of efficient use of blast-furnace gas. There certainly is room for great improvement along this line and it would not take much of a prophet to foresee the entire remodeling of our stove equipment in the next ten years.

WALTHER MATHESIUS, South Chicago, Ill. (communication to the Secretary*).—Referring to Mr. Johnson's question as to the amount of heat carried off with the molten metal and the slag, I beg to say that the figures used in my heat balance represent average values as generally quoted in German literature (cf. Ledebur: *Handbuch der Eisenhuettenkunde*, vol. ii, p. 258, 1906).

In order to determine experimentally these figures I made a number

* Received Mar. 25, 1915.

of calorimetric tests some years ago while working along similar lines of research at German blast-furnace plants. In nearly all cases I found that no regular foundry and Bessemer grades the results of my tests so closely approached the average values given in literature, that for the purpose of a general heat balance I felt justified in uniformly using the latter figures. Since neither chemical analysis nor the apparent physical condition of the iron and slag at Bessemer furnaces using Mesaba ores is decidedly different from the German practice on similar grades, I have no doubt that for the purpose of a general heat balance the German figures for the amount of heat in the iron and slag can be taken as representative also of American conditions.

Effect of Finishing Temperatures of Rails on Their Physical Properties and Microstructure

Discussion of the paper of W. R. SHIMER, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 99, March, 1915, pp. 557 to 585.

WILLIAM R. WEBSTER, Philadelphia, Pa.—I would like to ask Mr. Shimer how the finishing temperature of the H-beams and other beams rolled on the Gray mill at Bethlehem compare with the rolling temperatures of the rails referred to in this paper. Also, how the reductions in the rolling of the beams compare with the reductions in the rolling of the rails. I would also like to know if any drop tests were made on the rails referred to as having been allowed to cool and then annealed; in other words, what results, if any, were obtained from tests on rails after the internal strains of rolling had been removed; and in the matter of the rail blooms, I would like to ask if they are not often heated to a higher temperature than is desirable on account of the thin flanges in order to carry the heat through to the finishing pass.

The tensile strength of the steel in these rails is given as 125,000 lb. per square inch, which is almost as high as tire steel. If some of these blooms were rolled into rounds of about the same section as the head of the rail, would you not get better results from tension tests than those from the head of the rail? In other words, in rolling rounds, whether you would not be able to control the rolling and finishing temperatures better than is possible in the rail, owing to thin section of the rail flanges. Some years ago I had experience with the rolling temperatures of tire steel, running from about 127,000 to 133,000 tensile strength. These tires had the tensile strength and the deflection specified. One of the manufacturers had no trouble whatever, after a little experimenting, in meeting the conditions of the specification. Another manufacturer who made some of these tires came to the office and stated that he had tried tires from three heats of steel and could not meet the specifications and

wanted modifications. We examined the report of the chemistry of the steel, results he had achieved, and then asked him to go home and roll some more tires out of the same heats of steel and finish them as cold as he possibly could. He demurred to this, claiming that the tires had been rolled hot in order to meet the requirements. But he did roll the tires from the heats of steel he himself had condemned, and all of the tires met the specifications and were accepted.

I think the effect of finishing temperatures on rails has been merely started, and that all the factors entering into the problem will have to be considered before we can come to any definite conclusions. Will we in the end arrive at a better section of rail that will allow a better control of the reductions in rolling and finishing temperatures? Under present conditions we cannot give the steel fair treatment in rail sections with the wide, thin flanges.

HENRY M. HOWE, Bedford Hills, N. Y.—One thought in this connection is, perhaps, in view of such a high manganese content we need not expect so much influence from finishing temperature as we do in such material as Mr. Webster speaks of, because of the sluggishness of the metal in the presence of so much manganese. Very likely the question of finishing temperature may be less important in a 1 per cent. manganese material than in the material which Mr. Webster has been using, which is probably very much more sensitive to the finishing temperature.

* The probable key to the discrepancy is the influence of time. There is no doubt that we formerly attached too much importance to temperature and too little to time. Later investigations show that time is quite as important as temperature.

The opportunity for coarsening exists between the moment when the rail last undergoes sufficient deformation to break up the austenite grains and causes them to form anew and the moment when it finally cools below Ar_3 . The actual finishing temperature, as measured by the temperature at which the rail leaves the finishing pass, may be misleading, because the reduction in that pass and indeed in the leading pass and the one preceding may be insufficient to break up the existing austenite grains effectively.

Beyond this the rate of cooling of so thin a section of rail is likely to be so rapid as to cut down materially the opportunity for growth of the austenite grains. When a specimen is heated up to a given temperature, say $1,000^\circ$, held there and allowed to cool down slowly, the opportunity for grain growth includes first the period between leaving Ac_3 and reaching $1,000^\circ$, second the stay at $1,000^\circ$, third the time in cooling from $1,000^\circ$ to Ar_3 .

* Communication to the Secretary. Received Mar. 24, 1915.

What with hastening the cooling by water played on the rail, and with the rapid natural rate because of the thin section of the rail, the opportunity for coarsening is relatively short.

These conditions may explain why the influence of finishing temperature is less marked than might have been expected.

A. A. STEVENSON, Philadelphia, Pa.—The manganese content of the tires referred to by Mr. Webster was 0.70 to 0.80, sometimes 0.90.

G. K. BURGESS, Washington, D. C.—I am very glad to see that Mr. Shimer has been able to carry out these experiments and also has had the opportunity to publish them. One or two matters occur to me which suggest a possibility of the reason for some of the indeterminateness of the results. As I understand it, the blooms were taken from a reheating furnace and allowed to cool. I think there is some question in that operation as to the uniformity of the temperature throughout the bloom. Although Mr. Shimer has apparently, when he measured the temperatures on the surfaces of the finished rails, the characteristic condition which gives him an indication of the temperature of rolling, I think if it had been possible to control the temperature of his reheating furnace so that the bloom was unquestionably reheated throughout to the same temperature, then a source of indeterminateness would have been removed. However, there was, of course, some variation in temperature at least across a section of the rail, but in so far as these experiments prove anything, they prove that there is no very considerable effect due to finishing temperature on the properties of the rail.

Just a word regarding the microstructure. There is one inconsistency, at least it appears to me such, which I find difficult to understand. As it appears, the explanation of Mr. Shimer in placing the rails on the bed, he accounts for the variation in structure due to the difference in cooling, but his description of placing them on the beds, it seems to me, would make the *B* and the *E* rails receive identical cooling, yet in all cases the *B* and *E* rails show a very considerable difference in structure. Also, regarding any inconsistency between the shrinkage and finishing temperature, of course, when both are correctly taken, there should be no discrepancy there.

I think it is also of interest to mention the fact that within the last day or two a report was made by Mr. Wickhorst, of the American Railway Engineering Association, in which, so far as that report is conclusive, there is some slight evidence that a lower finishing temperature is advantageous in the properties of the rail. I think, however, that Mr. Wickhorst is not very emphatic on that statement.

Mr. Shimer's paper is a useful contribution to the subject, and it is hoped he will be able to execute the further experiments contemplated.

SAMUEL L. HOYT,* Minneapolis, Minn. (communication to the Secretary†).—The general conclusion to be drawn from this paper seems to be that the mechanical testing of rails, as now practised at the steel plant, fails to develop the true quality of steel rails.

The author states in the first sentence of the summary that "No appreciable difference in grain was found to indicate that the size or structure was governed by a difference in finishing temperatures." Such a statement would bear more weight if accompanied by a table giving the cell size of each specimen, at least for those specimens in which the free ferrite has separated out as a network.

The rails can be conveniently divided into two groups: (a) those with no free ferrite and (b) those with free ferrite. The rails under (a) consist chiefly of sorbite, so that the grain size is too small to be determined microscopically. The properties, so far as they are dependent on the grain size, would hardly be expected to vary appreciably. Any variation in the properties of these rails would more probably be due to difference in chemical composition, etc. Under (b) the grain size is likewise too small to be conveniently determined. There are, however, two important points in connection with the microstructure of these rails: the size of the cells, and the amount and distribution of the free ferrite contained in the network. These factors, under the conditions of the experiment, should vary with the finishing temperature.

For the sake of illustrating this point, I have compiled the following table, which gives the finishing temperatures of the *E* and *F* rails, tabulated consecutively, as well as the total number of cells contained within the microphotographs as closely as could be estimated, also tabulated consecutively. This table is hardly to be looked upon as being strictly accurate, but the results are so promising that the cell size, as measured by some one of the usual methods, might be advantageously added for the sake of comparison. Such a table would probably be helpful in connection with the finishing temperatures.

Cell Size of the E and F Rails vs. Finishing Temperature

No.	Finishing temperature, °F.	No.	Cells	No.	Cells
4	2,147	4 <i>E</i>	12	5 <i>F</i>	23
3	2,120	3 <i>E</i>	13	4 <i>F</i>	24
2	2,080	5 <i>E</i>	21	3 <i>F</i>	28
5	2,048	2 <i>E</i>	28	2 <i>F</i>	30
1	2,020	6 <i>E</i>	32	1 <i>F</i>	41
6	1,904	1 <i>E</i>	37	6 <i>F</i>	45
7	1,850	8 <i>E</i>	40	7 <i>F</i>	46
8	1,841	7 <i>E</i>	45	8 <i>F</i>	48
9	1,652	9 <i>E</i>	50	9 <i>F</i>	52

* Assistant Professor of Metallography, University of Minnesota.

† Received Mar. 12, 1915.

The author refers to the drop test as "the important test of the quality of the rail." This view is being somewhat discredited of late, on account of lack of discrimination. The mill tests are always on fresh steel and of very short duration. The service test of the rail, while in the track, is of comparatively long duration and on steel which is continually aging. A defect, sufficiently pronounced to cause a "rail failure," need not necessarily cause the rail to give abnormal results either in the testing machine or under the drop test, because the mechanism bringing about the failure in the two cases is different.

That the ordinary mill tests are more or less crude is substantiated by the conclusion of Burgess and Hadfield in their paper on the Comparison Tests of Rail Ingots that "The usual physical and chemical tests do not appear to give adequate measure of the quality of rails." The irregularities given in the tables of the present paper are further evidence of the crudities of these determinations of the quality of steel rails. Thus in the second half of the table on p. 564, rails *D*, *E*, and *F*, No. 4, have a lower carbon content than No. 9 of the same group, yet No. 4 is stronger and has a lower elongation and reduction of area than No. 9. (The value of 30.00 for No. 9 is evidently incorrect and should be 26.66.) This shows there must be factors other than small variations in the carbon content which have an influence on these results, and which are not brought out in the tests.

The higher finishing temperature and lower carbon content of No. 4 as compared to No. 9 should give results exactly the opposite of these. The above could be overlooked if the absolute value of the material alone were desired, but when relative values are needed, all the variables save one (finishing temperature) should be kept constant. It is apparent that, in ordinary plant practice, even though carefully conducted, there are too many variable factors, the influences of which are sufficient to obscure the results sought. Even at that it is rather difficult to understand the author's meaning in "all the tensile and drop tests recorded are good, and *practically identical*."¹ For example, the elastic limits (yield points?) vary by about 16 per cent. in the tables given, etc.

The author's statement to the effect that the strains set up during the reduction of the ingot have an effect on the properties of the finished section is at variance with the generally accepted theory that (with excessive reduction excepted) it is the grain and cell size, produced during the hot rolling and on cooling, that influences the properties. The strains set up during "hot rolling," if one chooses to consider it this way, are relieved immediately (or nearly so) during the recrystallization which accompanies the reduction in grain (crystal) size.

¹ The italics are mine.

Sound Steel Ingots and Rails

Discussion of the paper of GEORGE K. BURGESS and SIR ROBERT A. HADFIELD, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 455 to 468.

ALBERT SAUVEUR, Cambridge, Mass.—There is a definiteness and finality in the investigations that have been conducted by the Bureau of Standards and in the method of presenting the results that is most satisfactory and refreshing. A year ago Dr. Burgess and his co-workers investigated the question of critical points in carbonless iron, and they settled the question once for all. No one can now claim that the thermal point A_2 in carbonless iron does not exist.

At the last meeting of this Institute in Pittsburgh, Dr. Burgess showed the worthlessness of the shrinkage clause. To my mind after his demonstration no one can again claim that the shrinkage clause renders any service. He also showed us that it was a practicable proposition to use pyrometers in steel work in order to ascertain the finishing temperatures of rails. No one can again come before this body and make the claim that it is not a practical proposition.

And now Dr. Burgess shows us that steel ingots can be made in a commercial way, practically free from pipe, from segregation, and from blow holes. It places again before us in a very clear manner the question of piped ingots and of segregation, a question which has been debated for 20 years or more. I know very well the attitude of the standpatters in the matter. They claim that most of the pipe is removed in the discard, and that if a little is left it is so thoroughly welded that it does no harm. I for one am not convinced of the soundness of the argument. I do not believe there is any one present who would claim that, given an ingot free from pipe, free from segregation, and free from blow holes, better rails will not be made out of that ingot than out of the ordinary piped or segregated metal which is used by almost all of our steel mills. In my opinion, if we give our sanction to the casting of steel in the shape of piped or segregated steel ingots, we are guilty. If we give our sanction to the rolling of rails from such metal, we are guilty. It is our duty as experts, as steel makers, and as steel consumers to make and to use the best steel obtainable on a commercial basis, for the manufacture of rails.

Now, we have been shown that sound ingots can be produced from which superior rails can be made. If we do not insist upon these sound ingots, then we fail in our duty. I think that the art of casting steel ingots has advanced to such a point that the casting of steel ingots piped and segregated ought no longer to be tolerated.

I do not mean to say that the Hadfield process is the only one that

will do it. Indeed, I know that there are other processes, and that at some mills methods are used which will practically prevent the casting of piped and segregated metal, but I claim that the practice should become general.

HENRY M. HOWE, Bedford Hills, N. Y.—I think the ethical question now comes up—is it not incumbent upon us in every reasonable way to bring every reasonable pressure to prevent the continuation of peril to the lives and safety of passengers on railways from the use of piped and segregated rails?

* Can the authors tell us in how great a proportion of cases the suppression of segregation and piping is as thorough as it is in the cases which they show? Or can they tell us how large their experimental basis of observation is, to the end that we may form an opinion as to how large the proportion of cases is in which these very great benefits may confidently be expected?

I of course have in mind the experience which so often has come to us, that an apparently very great improvement occurs in certain cases, but is not reproduced regularly in all, or is reproduced in only a certain proportion of cases.

HENRY D. HIBBARD, Plainfield, N. J.—Dr. Burgess has stated that ingots cast by the Hadfield method are always right, but it should be understood that no method of casting will make good ingots unless the steel is right before it goes into the molds. Beside having the proper composition as to desired ingredients it must have full tendency to pipe, and all impurities within permissible limits. These impurities are the soluble phosphide and sulphide of iron, the insoluble oxides, silicates and sulphides of iron and manganese, and the gases, carbonic oxide and hydrogen. When rail steel meeting these requirements is cast into pipeless ingots, good, sound, serviceable, trustworthy rails can be made therefrom.

JOSEPH W. RICHARDS, So. Bethlehem, Pa.—I think that both this paper and the preceding one should have a short paragraph inserted in the paper saying how the temperatures were determined, the kind of apparatus and the way it was used, because probably these results will be duplicated by others, and there may be misunderstandings, because of using different methods of measuring the temperature.

F. W. WOOD, Sparrows Point, Md.—(communication to the Secretary†).—While agreeing fully that sound ingots with a minimum of segregation can be produced by Sir Robert Hadfield's method of casting, if properly conducted, it is an open question to what extent the total of rail failures in service would be reduced by its general adoption.

* Communication to the Secretary. Received Mar. 25, 1915.

† Received Feb. 13, 1915.

The statement that ingots so cast will give rails free from flaws of all kinds, is assumed to refer to interior flaws.

Obviously, exterior seams, the result of breaks in the skin of the ingot from shrinkage while in the mold or from the early passes in the blooming rolls, will not be affected. Likewise, inclusions of slag or other foreign matter, near the outside of the ingot, presumably the starting points of split heads, may be expected to be as numerous as in ingots cast in the usual way.

The obscure and very dangerous type of failure, the transverse fissure, which develops in rails from all parts of the ingot, has not been connected with piping or segregation, the two irregularities the Hadfield process is claimed to remedy.

Records from railroads with heavy traffic show breakages of flanges and through the entire section of the rail to be as numerous in rails from the lower two thirds of the ingot as from the upper, and consequently not mainly due to pipes or irregularity of composition.

It is not the purpose of these comments to question the desirability of sound and non-segregated ingots as a factor in securing the best possible service, but rather as supporting the writer's opinion that any positive relief from rail failures, especially of dangerous types, must be looked for in other directions.

GEORGE K. BURGESS.—I will say, in regard to Professor Richards's statement, as a matter of record, that the temperature measurements in this communication were made exactly in the same way as the temperature measurements referred to in the previous communication, on finishing temperature of rails, which I presented at the last meeting.

Of course, the statements made by Mr. Hibbard, as to first having sound steel, are unquestionably acceptable from all points of view.

The several factors contributing to the production of unsound finished products, to which Mr. Wood calls attention, are undoubtedly vital, but the existence of these various imperfections should serve as an incentive to eliminate each cause of unsoundness as opportunity offers.

HENRY M. HOWE (communication to the Secretary*).—On p. 459 it is said that "the comparison ingot if rolled into rails should have about 50 per cent. discard." This seems to require some explanation. A discard of a little over 20 per cent. would have removed all the metal containing more than 0.104 phosphorus, and the remaining nearly 80 per cent. would have had no metal with more than 0.096 per cent. except for a rather short distance along its axis, where it rises to 0.104 per cent. With an average phosphorus content of 0.09, an enrichment in the very axis up to 0.104 certainly is not serious.

An examination of the segregation indicates that the advantage of the

* Received Feb. 24, 1915.

Hadfield casting is not in a reduction of the degree of segregation but in confining segregation to the very top of the ingot, as is natural. For instance, the greatest Hadfield enrichment in carbon in Fig. 4 is from 0.59 to 0.79, a gain of 0.20, or 34 per cent. The enrichment in carbon of the comparison ingot is from 0.46 to 0.59, a gain of 0.13, or 28 per cent.

But there is this difference, that the Hadfield 0.79 is confined to about 5 per cent. of the upper part of the ingot, whereas the 0.59 of the comparison ingot is somewhere about 15 per cent. from the top.

The hardness numbers in Table III are not so favorable to the Hadfield metal as might have been hoped. The A rail of the No. 3 ingot has a greater average deviation than any of the rails with which it is compared, and the next greatest deviation comes in the Hadfield A4 rail, 10 per cent.

A result of interest is the comparison of the piping steel, No. 6, and the rising steel, No. 7, in Tables III and VI. The piping steel has a smaller deviation in hardness than the rising, and a much smaller deviation in carbon content, namely, 0.528 to 0.584, or 0.056, whereas the rising steel has a deviation from 0.550 to 0.648, or 0.098, nearly twice as much.

* The belief that the grain size reached increases with the total time available in heating, at heat, and in cooling, is supported by the data of Burgess and his collaborators,¹ although they put a different interpretation on them. They find that the average number of grains per square inch of 90-lb. and 100-lb. rails is 24,000, whereas that of 72-lb. and 75-lb. rails finished at the same temperature is 42,800. Thus the thinner and more rapidly cooling rails have developed grains only about half as large as those of the larger and hence more slowly cooling rails.

Further, they find the average number of grains per square inch for all their rails is only 26,200 for the thick and slowly cooling head, against 44,800 for the web and 36,900 for the base. These differences are no doubt referable in part to the higher finishing temperature of the head, but they certainly support the belief that the necessarily slower cooling of the head also has favored grain growth.

The comparison of the web and base is especially interesting. As they leave the rolls the base certainly ought not to be materially hotter than the web, and hence, if the grain size depended solely on the finishing temperature, its grain size ought not to be greater than that of the web. But on the hot bed the cooling of the web is likely to be materially faster than that of the base, for the reason that the base of each rail rests against the thick hot head of the next, whereas the web is exposed freely to the air on both sides. Hence the smaller grain size of the web than of the base seems clearly due to the more rapid cooling of the web, the shorter time available for grain growth.

* Received Mar. 27, 1915.

¹ *Technologic Paper No. 38, U. S. Bureau of Standards, Observations on Finishing Temperatures and Properties of Rails, p. 41 (1914).*

Are the Deformation Lines in Manganese Steel Twins or Slip Bands?

Discussion of the paper of HENRY M. HOWE and ARTHUR G. LEVY, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 99, March, 1915, pp. 587 to 600.

J. E. STEAD, Middlesbrough, England (communication to the Secretary*).—As I have not personally studied the question of quenched and distorted manganese steels I cannot give any information of my own on the subject.

The question as to whether certain indications in strained manganese steel are "slip bands" or "twins" might be settled, I think, by heat tinting—a method I have applied with success to certain meteorites and other alpha irons in which there were undoubted twins. After polishing and very slightly etching with picric acid in alcohol, and then "heat tinting," the colors assumed by the twinned strata and the adjoining metal are different. The reason of this, as I have proved repeatedly, is that different faces of the iron crystals are differently attacked by oxidizing and other reagents, and consequently some of the faces take temper oxidation tints more rapidly than others. This simple method enables one to determine with certainty whether the "twins" are actually twins and differently oriented from the bed in which they are found. After correspondence with Dr. Howe, I think we both agree that Neumann lines should not be called "lines" but "bands" or "strata," preferably the latter term, as I have found that they are more or less continuous throughout the crystals, and it would be just as improper to call the edges of cementite plate by the term "lines" as to call stratified twins by the same term.

The observation that on annealing strained austenite manganese steel, cementite settles out along the slip planes, is most interesting and is a quite original observation.

* Received Mar. 26, 1915.

Structure and Hysteresis Loss in Medium-Carbon Steel

Discussion of the paper of F. C. LANGENBERG and R. G. WEBBER, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 291 to 300.

HENRY M. HOWE, Bedford Hills, N. Y.—The results here given are of great interest and are welcome as forming another rivet to increase the stability of our present theories of the constitution and properties of steel.

The important discovery is that the hysteresis increases with the fineness of the structure, that is to say with the extent of contact between the particles of ferrite and cementite. All of the specimens except No. 4 consist of ferrite and cementite of different degrees of intimacy of admixture.

If hysteresis represents something akin to surface friction, which retards the rotation of the ferrite on the application and release of the magnetizing force, then it is only natural that it should increase as it does with the intimacy of admixture, because it is at the surface of contact of a given particle of ferrite with the neighboring particles of cementite that this friction is to be expected, and of course the extent of surface must increase with the intimacy of admixture.

The somewhat greater hardness of specimen 4 than of the others may perhaps be referred to the principle discussed in my paper, *Why Does Lag Increase with the Temperature from Which Cooling Starts?*¹ The furnace-cooled specimens were cooled so slowly that the influence of the initial temperature of heating is effaced. But in the air-cooled specimens we have the suggestion that the high temperature, 1,000°, from which the cooling of No. 4 started has prevented the completion of the transformation, with the result that the metal is slightly harder than any of the others.

The difference is not to be explained by difference in the fineness of structure, because the whole range of fineness from No. 1 to No. 6 is accompanied by no change in hardness, the only change being an appreciable increase of hardness following the air cooling from 1,000°.

¹ *Trans.*, xlv, 516 (1913).

German and Other Sources of Potash Supply

Discussion of the paper of CHARLES H. MACDOWELL, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 103 to 114.

GEORGE S. RICE, Pittsburgh, Pa.—I am especially interested in the mining side of this important question. In 1911, when in Europe investigating mining matters, giving special attention to subsidence due to mining, for there is a very big problem of that kind, as many of you know, in the Pennsylvania anthracite region, I visited Stassfurt, which is one of the chief centers of potash mining, and which was pointed out as the place to see surface subsidence. We found the town much askew—some of the streets having sunk from 15 to 30 ft., leaving many ancient buildings pitching at various angles. One old church, 500 or 600 years old, has been sunk about 20 ft., cracking it badly.

I understand that this condition came about because in early mining days they were careless about leaving proper supports and used no filling or packing. The Stassfurt mines work along a buried anticline the sides of which pitch at 45° on the average. Through this careless mining they had a break of the hanging wall which let in water from the overlying marls and sands. The water rapidly dissolved the salts in the pillars, and spreading through the entire chain of mines, caused their complete loss, with serious trouble for the old city. They had to start new mines some distance away, and take care to prevent roof caves.

The packing system is of interest. They were driving large rooms into the rock-salt foot wall and using the rock salt as dry filling.

The application of hydraulic filling, using a saturated solution, was under contemplation at some plant nearby. Whether or not the plan was carried out I do not know. I was reminded of the plan by Mr. MacDowell's remarking on the use of a saturated salt solution in the core drilling.

His reference to the apparent breakdown of the potash syndicate raises an interesting question. We have had the German system of syndicates pointed out as perhaps one means of solving our troubles in the soft-coal mining industry due to overproduction. It seems to have worked out admirably in the case of the Westphalian coal-mining syndicate and also in the syndicate of Upper Silesia; but apparently the syndicate plan did not work in the potash industry.

One thing that was possibly a helpful factor in the success of the Westphalian syndicate was that the Prussian government decided some years ago, so I understand, to appropriate all future developed extensions of the coal fields to the north of the known field; this probably prevented

that bad feature of the American coal industry, the promiscuous opening of mines.

I hope Mr. MacDowell can give us a little light on why the potash syndicate failed.

CHARLES H. MACDOWELL.—The question of controlling prices by sales through a syndicate has several sides to it. In Germany, almost since the beginning of the industry, they have tried to restrict the number of new mines. They called in an old coal-mining law which required two shafts—a safety measure. The owners of the mines simply split up the holdings, put down a second shaft, formed a new mining, and asked for additional quotas. The great trouble over there has been to curtail the activities of the man who has mining shares or rights to sell. In order to satisfy the large number of mines they had to put up the price, causing new companies to be floated so they pyramided. Then the new monopoly law made matters worse.

At the time this monopoly law was introduced there were 52 mines, and now there are 160. The increase came largely from the fact that the people felt the profits must be large under a government monopoly.

I do not think these large selling cartels can be made successful unless there is some method of limiting the number of mines. The business is increased by propaganda; but the increase does not keep pace with the willingness of people to put money in new properties. There is always a fight as to percentages, and one group works against another. Unless there is some way of controlling output, I question whether the syndicate method is a successful one.

It is not wise to mine more than is needed, and if there are too many producers, wasteful mining methods are sure to be resorted to, to cheapen mining costs. Output control and fair prices are necessary if true conservation is desired.

V. C. GRUBNAU.—Has any process been discovered for obtaining the potash salts from wool scourings?

MR. MACDOWELL.—Yes, a fair tonnage is obtained in Germany and other countries from wool scourings.

Another suggested source of potash is feldspar, concerning which Dr. A. S. Cushman has recently published an article. He estimates the cost of producing muriate of potash from feldspar as \$30 per ton. That is a high cost, for the contract we had with the Germans, over which our trouble arose, was based on a price of \$19 per ton; and some of the mines, as I have stated, claimed to have made it for \$10. Unless war-time conditions continue, I think that muriate of potash from feldspar will not have much chance.

White-Burning Clays of the Southern Appalachian States

Discussion of the paper of JOEL H. WATKINS, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 391 to 411.

H. RIES, Ithaca, N. Y.—It is interesting to contemplate to what extent the American pottery industry would be affected in case the English supply of china clay were shut off for a considerable period of time. The English deposits are very large, and are worked on a most extensive scale. They are more easily and probably more cheaply worked than our American deposits, which are of vein-like nature. The American producer of washed clay is not always as careful as the foreign miner and washer of this material, although he has improved in recent years.

Our American washing plants are, furthermore, not as economical of material as it seems to me they might be, for the reason that more or less of the finest clay is often allowed to overflow, due to insufficient settling tank capacity. This fact is often shown by the milky character of the streams into which the waste water is discharged.

The Origin of the Louisiana and East Texas Salines

Discussion of the paper of EDWARD G. NORTON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 97, January, 1915, pp. 93 to 102.

G. D. HARRIS, Ithaca, N. Y. (communication to the Secretary *).—Since I have on several occasions¹ expressed at length my general views on this interesting subject, it seems fitting that I endeavor here to comment as briefly as possible on a few points suggested by the paper. Doubtless you all know of the rather extensive publications of Lachmann on mid-European salines and his scarcely successful attempt to make the phenomena presented by our domes fit into his "Exzeme" theory based on studies of European phenomena.

This paper is a most welcome indication that scientific America is very much alive to the difficulties that have been encountered in attempts already made to explain the origin of the salt domes in the lower Mississippi embayment region. Praiseworthy, too, is the attempt to explain these phenomena by agencies still in operation. I may, perhaps, be permitted to state here that a former student of mine, and co-worker in Louisiana, A. C. Veatch, is now interested in the Algerian region, where it would seem similar dome-phenomena are to be found, and hopes may well

* Received Mar. 6, 1915.

¹ *Economic Geology*, vol. iv, No. 1, pp. 12 to 34 (1907).

Bulletin No. 7, *Louisiana Geological Survey*, pp. 59 to 83 (1907).

Bulletin No. 429, *U. S. Geological Survey*, pp. 6 to 12 (1912).

Popular Science Monthly, January, 1913, pp. 187 to 191.

be entertained that this shrewd and zealous geologist will some day furnish additional light on the subject from that distant land. As for myself, I have had occasion to examine many of the domes recently in great detail, and have had a student working in the laboratory along the line of precipitation from solution through decrease in temperature, and hence may say definitely that such precipitation, with the formation of salt crystals, is quickly and copiously obtained. The necessity of assuming surface evaporation in the formation of these crystalline, saline masses seems therefore to be no longer pressing. The force exerted by growing salt crystals is next to be taken up.

I will limit my remarks to two points in the theory before us. First, the assumption of volcanic activity. Having made complete trigonometrical and magnetic surveys about two representative domes where salt masses are known to occur I have found that the local deflections observable are within the limits of instrumental error, about 2 min. The marked deflections about the nearest igneous matter, the peridotites of southwest Arkansas, I have personally measured; those about Hot Springs are too well known to need commenting on here. Local-heated nuclei or even cold plutonic plugs would almost certainly have made themselves known, if present, by these investigations. The "hot" waters so far observed in any of the great flows from wells put down in domes or off of domes have revealed scarcely an increase of 1° F. to every 50 ft. of descent. I have seen nothing suggesting local hot rocks below. Again, the use of such terms as travertine and "calcareous sinter" for the hard blue and white highly crystalline limestone (called "marble" locally) I believe is apt to produce a wrong impression. That there may be a slight solution and occasional redeposition about the peripheries of these saline domes no one can doubt, though the redeposition of chloride of sodium in present climatic conditions, so far as a permanent rock-forming process is concerned, is entirely out of the question. The rocks referred to as light-gray limestone at Drake's and elsewhere are more arenaceous than calcareous, as determined by subsequent examination, and are of the ordinary sedimentary type, having nothing in common with dome rock, as intimated in this paper. The fact that they contain leaves is therefore not strange, nor does it imply contemporaneity of dome formation and sedimentation. Personally I would be as sanguine of finding fossils in any batholith as in any dome rock.

Second, as to quaquaversal structure. This to me is the key to the whole situation. And it is with distinct regret that I find this all-important subject dismissed with no new field observations and with general remarks covering less than four lines. Naturally one might believe *a priori* that after a dome mass had formed, there might be some slight settling down about the same, giving rise to a certain amount of dragging and bending of the edges of bedding planes against the dome mass. But

field observations will allow of being treated with no such academic and slight consideration. How can "faulting and downward displacements" explain the presence of Cretaceous beds lying upon and flanking a local, small, circular dome that peers out through Lower Claiborne deposits 2,000 ft. above the undisturbed Cretaceous deposits? How may they be so crushed, compressed, and hardened till all bedding planes are obliterated and systems of cleavage or joint planes are established by simply "faulting downward"? Perhaps I can make the difference between the author's and my views on this subject of relative motion clear by means of a homely comparison. Suppose on entering your newly completed house you notice the carpenters have left the point of a nail protruding through the floor. Being of a scientific turn of mind you may investigate the matter closely, and finding the woody matter broken upward all about the nail in quaquaversal manner, conclude either that some careless "hand" had driven the same up from below, or you could possibly imagine that the nail had remained *in situ* and the whole floor had sunk about it. So, too, you may imagine the whole rock floor of northwest Louisiana as having sunk 2,000 ft. about these local circular salt spikes, but I still prefer to think of the latter as having been driven up from below according to the theory discussed under the references herewith given

Experiments on the Flow of Sand and Water through Spigots.

Discussion of the paper of R. H. RICHARDS and BOYD DUDLEY, JR., presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 97, January, 1915, pp. 67 to 72.

R. H. RICHARDS, Boston, Mass.—I believe that the manner of working for efficiency in almost all lines is to study the conditions of highest efficiency, and then to measure what is being done. We can then see how near we have approached the highest efficiency. Spigots in practice often use many times the water theoretically needed. There are places where that is necessary for safety reasons. There are, however, places where it is unnecessary, and in these places it seems to me that this work of Mr. Dudley's will be a great help to the mill man to arrive at the greatest economy of water and highest efficiency.

ALBERT R. LEDOUX, New York, N. Y.—A good many years ago I received a telegram from that distinguished engineer, Hamilton Smith, which simply said in substance: "I have discovered that liquids will flow faster from a square orifice than from a round orifice of the same area, under the same head." I wondered why he sent me the telegram, but some time afterward he told me that he had been making experiments along that line in Montana; that he thought the discovery was interesting

and that I would be pleased to have early information of it; and that he was going to make it public when his tests were completed.

For a year or two after that, I watched the technical journals to see if he had ever published anything on the subject, but I have never seen any printed statement along this line. A few years afterward—as we all know—Mr. Hamilton Smith died.

I mentioned this fact to learn if the authors' experiments verify Mr. Smith's statement; or if they have found a paper by Mr. Smith covering this matter, which I may have overlooked.

BOYD DUDLEY, JR.—Perhaps some of the information contained in this paper was originally due to Mr. Smith. I do not know. However, it comes from Merriam's *Treatise on Hydraulics*, and this is given—"For orifices having a sharp inner edge, which is alone touched by the water, C equals 0.59 to 0.63, being greater for low than for high heads, greater for rectangles than for squares, and greater for squares than for circles." That covers the point, I believe, that has been raised, but I do not know just to whom the credit is due.

Development of the Butchart Riffle System at Morenci

Discussion of the paper of DAVID COLE, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 431 to 444.

R. H. RICHARDS, Boston, Mass.—The Butchart riffle seems to violate, according to our preconceived notions, some of the very essential principles of the Wilfley riffle, and therefore ought not to succeed, but it does. On this account it must conform to other very important principles, or it would not succeed. It would be interesting to know what these principles are which enable it to succeed.

E. P. MATHEWSON, Butte, Mont.—I cannot explain the principles on which this riffle acts, because I have not given it personal attention. All I can say is that I first saw the riffle in operation several years ago at the Steptoe Valley mill in Nevada, and I was told that a man had been around there guaranteeing that these riffles would make the table do twice as much work and do it better than with the ordinary Wilfley riffles. I heard nothing further about it for a long time.

I heard about a year ago, from David Cole, the author of this paper, that the riffles were a success, and that he was using them at Morenci, and if we were interested he would send the inventor to show us how they worked. As we knew of David Cole's reputation, we invited Mr. Butchart to come up and show us how his riffles worked. We did not have sufficient confidence in the invention to guarantee Mr. Butchart his traveling expenses, but he came up entirely at his own expense and put in a set of the riffles on one table. It was known as a roughing table in our No.

1 section. We put it to work, tried it, and had it carefully tested by our testing department, and we found on roughing work it was a grand success, that we could get more tonnage and better work done by this device. We were encouraged by Mr. Butchart to try it on classified feed, and there we found that it did not do such good work, that it was considered no improvement with such feed. We finally made an arrangement with Mr. Butchart to put this device on all the *roughing* tables we had at that time in our mill, which were nine or ten. About the time we had this introduced, we made a new design for our mill, and a new flow sheet. We adopted the Butchart riffle for the entire mill, for our *roughing* tables, and these will be the only tables of the Wilfley type we will have in our mill.

DAVID COLE, Morenci, Ariz. (communication to the Secretary*).—Mr. Mathewson is substantially correct in his remarks, and in way of supplementing his statement I would add that this riffle system was first developed in an experimental way by Mr. Butchart in Los Angeles. After he completed his experiments there he came to Morenci and made the first practical application of the riffle in the No. 6 concentrator of the Arizona Copper Co. This was in the latter part of 1911. He carried on a long series of experiments at that time and these led to the adoption of the riffles by Mr. Carmichael for the 22 tables then in use. Experiments with the new riffle system were also carried on at the same time in the Detroit Copper Co.'s mill here. One or more tables were riffled for experimental purposes and operations were carried on under the supervision of the company's metallurgist, but the Detroit company did not adopt the riffle until two years later, after it had been perfected in the Arizona Copper mill, as mentioned in my paper.

Subsequent to these original applications of the riffles at Morenci, Mr. Butchart carried on some experiments in the Steptoe plant of the Nevada Consolidated, and I understand that one section of the mill was riffled with his system. This worked so well that the company purchased the right to use the riffle in its entire milling plant.

All of the tables equipped with the Butchart system of riffles up to the time above referred to carried thin strips, which were applied to tables handling relatively fine feed; that is, the field of the new system of riffling was not extended beyond that of the field of the regular Wilfley riffle. No attempt had been made to handle feeds coarser than had been found to be adapted to the Wilfley system. It was found at that time that the new system would handle larger tonnages than the Wilfley and, as mentioned in my paper, the tables were much more easily looked after, but the ability to handle very large tonnages of very coarse, unclassified feeds had not at that time been developed.

When I assumed the responsibility of remodeling the Arizona Copper

* Received Mar. 29, 1915.

Co.'s No. 6 concentrator about $2\frac{1}{2}$ years ago, I found the riffle in use in the mill and was favorably impressed with its work. I thought I saw possibilities of improvements and I advised the company to purchase the right to use the riffle in all of its table work instead of only the 22, and this was done. Mr. Butchart then came to Morenci and remained several months working out the improvements to the successful issue mentioned in my paper.

In July, 1914, I visited Montana with reference to advising the Anaconda company as to changes contemplated in their Washoe concentrator, and while at Anaconda I put a set of Butchart riffles on a Wilfley table doing roughing work in the No. 1 section, which had been remodeled so as to use the "Great Falls" system of concentration, described by Mr. Wiggin in his paper read at the Butte meeting. I operated this table for several days with excellent results. After returning home I communicated this fact to Mr. Butchart, who expressed a desire to go to Anaconda at his own expense to see the table in operation there, and, if the management should desire him to do so, to carry on further experiments. This led to the arrangement and results mentioned by Mr. Mathewson.

The new system of riffling is not well adapted to handle classified feeds. Hydraulic classification, carried out in the conventional manner common in concentrating practice, does not separate the feeds properly for the Butchart riffle. The coarser the feed the more necessary it is to have mixed sizes. For primary work not making tailings no classification or desliming is necessary, and for conventional table sizes when making finished tails it is, as I have pointed out in the paper, only necessary to remove the very finest sand together with the colloidal slime, for the table does the classifying after this and it requires the large range of sizes in the feed in order to hold the mineral and prevent losses; for all of the fine mineral would not have been taken out by hydraulic classification—some of it would go with the table feed and would not be held on the table except in the presence of suitable size sand.

Hydraulic classification is in fact rough concentration and hydraulic classifiers are rough concentrators, the use of which is very largely displaced by the use of the Butchart system of riffles.

Concentration engineers and mill men are slow to accept this feature of the situation because they have been so thoroughly drilled in the necessity for screen sizing and hydraulic classification that it has become a sort of religion in the trade and to change it is almost heresy. They regard the old order of things as law, but the operation of the Butchart riffle proves these ideas to be largely erroneous.

Very great improvements have been made in the past year in crushing, sizing, elevating, and concentrating machines. These improvements have had for their primary object the reduction of all divisions of costs through the reduction of stages, thus simplifying and cheapening the

concentration process. The Butchart riffle has aided materially in carrying out this program and with the marked improvements in crushing and screening machinery above mentioned we are now able to dispense with a large number of the operations previously thought absolutely essential. We can now crush the ore from run of mine to the first Butchart table size by one pass through a primary crusher plus one pass through a ball mill running wet. We can put the pulp so prepared direct

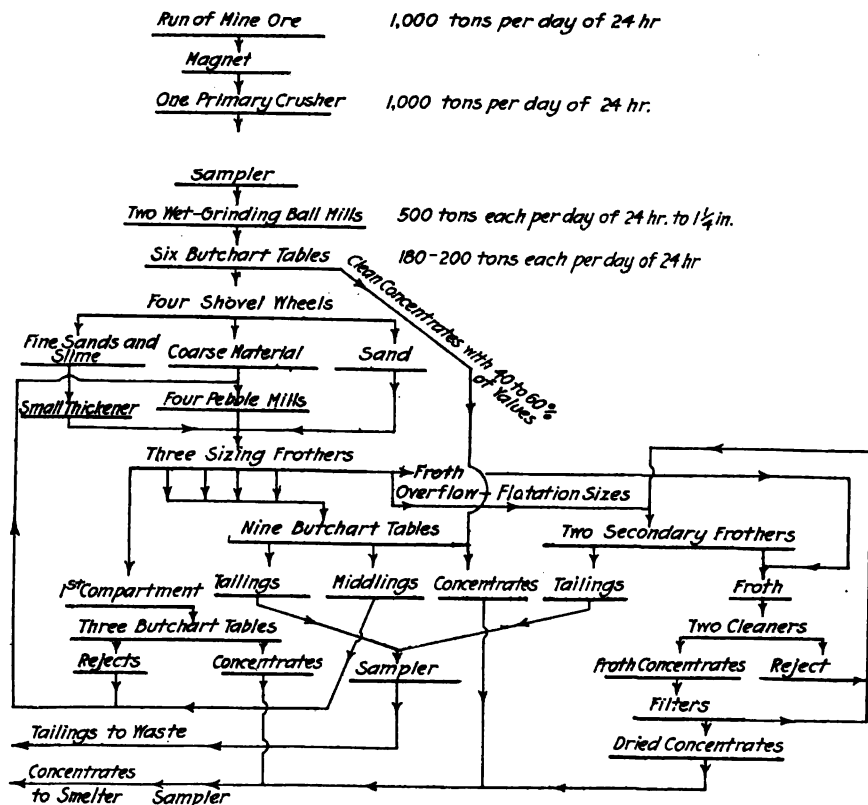


FIG. 6.—FLOW SHEET FOR LOW-RATIO COPPER ORES.

from the ball mill on Butchart tables *without any classification or deliming*. We can separate on these tables, in a condition to go directly to the smelter, a very large percentage of the minerals present. This concentrate will contain sizes from 1/4 in. to 200 mesh, in a clean condition. The colloidal copper will pass over the side of the machine with the slime rejections and if desirable may be separated by suitable mechanical classifiers so as to by-pass the final crushing machine. Demonstrations now being carried on here indicate strongly that the final regrinding mill will be most efficient and desirable when run in a closed circuit with a

frothing sizer, the latter having capacity to complete three important operations: first, pass the colloidal copper sulphide, the so-called slime or floured flake copper, over the top of the machine with the froth; second, remove the oversize which has escaped from the regrinder, in a condition to be returned to same; third, prepare the sands already sufficiently fine so that the feed will be ideal for table concentration—this will be the *natural* result when the slimed copper is removed by the frothing and the

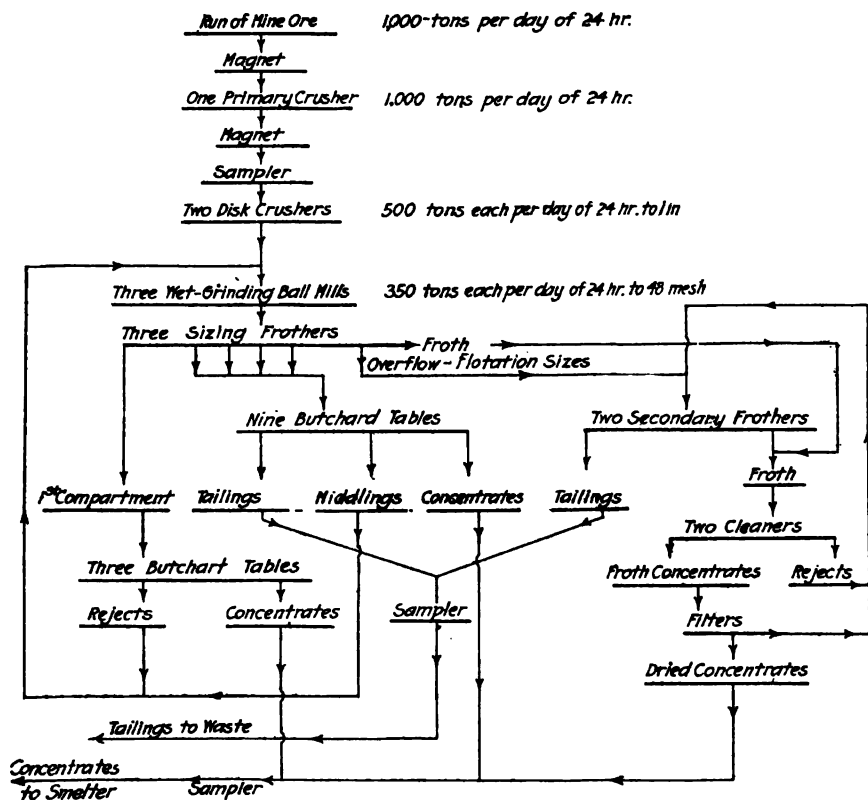


FIG. 7.—FLOW SHEET FOR HIGH-RATIO COPPER ORES.

flotation sizes are removed with the overflow of the machine. This overflow will be treated by further frothing if necessary. With this arrangement the operator of the frother has control of the size at which the pulp is allowed to leave the closed circuit, and the advantage in allowing the mill to make a quantity of oversize will be obvious, for when held down to grinding all of their product to pass the limiting screen without oversize the capacity of these grinders is greatly reduced.

In the above arrangement there is no colloidal or floured flake sulphide in the feed prepared for the tables—this all goes over the top with the

froth or with the flotation sizes—and thus the slime “bugaboo” is removed from the mill man’s list of troubles.

These improvements are important and far reaching and mean that we can easily double the capacity, under the same roof, of most copper concentrators now in operation and can construct new plants in much less space and for a very much less capital expenditure than was formerly required. I do not hesitate to say that however much we dislike to do it, we will undoubtedly abandon the old and complex system, with its multiplicity of screens, hydraulic classifiers, settling tanks and slimers, for with the addition of flotation treatment of the slime together with the great improvement in milling mechanism above referred, to very greatly simplified flow sheets (see Figs. 6 and 7) and very simple methods of treatment for the concentration of ore, particularly of copper ores, are demonstrated as now possible, and, in the opinion of the writer, these methods will quickly be recognized and adopted.

The Boulder Batholith of Montana

Discussion of the paper of PAUL BILLINGSLEY, presented at the New York meeting February, 1915, and printed in *Bulletin* No. 97, January, 1915, pp. 31 to 47.

JAMES F. KEMP, New York, N. Y.—Mr. Billingsley has spoken so concisely that we may not realize the many important topics which he has passed in review and upon which he has thrown much illumination. The exact determination of the age of the granite, itself, is one. If we were to look over a geological map containing the region of Butte and 25 years old, we would find this coarsely crystalline mass assigned, as were others of its kind, to the pre-Cambrian, or as we used to say, to the Archean. In the course of time it was found to be later. The *Butte Special Folio*, 1907, described it as post-Carboniferous and possibly post-Laramie. Mr. Weed in *Professional Paper No. 74 of the U. S. Geological Survey* (1912) placed it as probably Niocene because of its relations with the andesite, which was believed to be Eocene. Mr. Billingsley now by a very neat demonstration circumscribes it more definitely and we recognize that it must be latest Cretaceous or earliest Eocene. We see, however, that extensive field observations, ably correlated, were necessary to establish the conclusion.

For some years all familiar with Butte have known that there was a central area the veins of which carried copper ores in a quartz gangue; and that outside this area there were veins carrying silver ores in a gangue of manganese minerals. Mr. Sales’s able and detailed paper, read at the Montana meeting, August, 1913, not only corroborated these relations, but emphasized the transitional character of the intermediate or passage zone. Mr. Billingsley is now able to show from observations at increasing depths, the tendency of the copper ores to appear deep down in the veins

once regarded as silver-manganese veins. There is some reason to think that the copper ores run in beneath the silver veins as we pass outwardly from the copper center.

In some of the smaller camps away from Butte, apparently the veins carry galena above, base-metal ores beneath, and lean copper-bearing pyrite in depth. As to the cause of vertical distributions of this character, many of us have pondered seriously in later years. The explanation that a complex, heated, uprising solutions first lose their lean copper-bearing pyrite, as heat diminishes and pressure wanes; next their zinc, and lastly their galena, seems better than that secondary enrichment or rearrangement in an originally complex, uniform vein has left the galena above, concentrated the zinc below, and the copper-bearing pyrite lowest of all.

Mr. Billingsley has brought out many interesting points as to the relation of the veins when first filled to the original top of the granite mass. They do not seem to have been greatly decapitated by erosion.

Upon one other phase of the batholith additional discussion in a more elaborate treatment would be welcome. Apparently the paper supports the view that the great granite mass made a way for itself by breaking down and absorbing into itself the overlying rock. The process is technically called stoping, the miner's term having been adopted by the geologist. As many of us know, the view has been especially supported by Prof. R. A. Daly of Harvard. We are forced to assume that the detached blocks sink into the molten mass and are absorbed or "digested." We would be glad to know if Mr. Billingsley and his co-worker, Mr. Grimes, have been able to detect anywhere a change in the chemical composition or mineralogy of the granite, from this absorption of outside matter. Is it anywhere visibly more siliceous from absorbed quartzites or more basic from limestone? No more interesting question is before geologists to-day than this one of the upward stoping of batholiths and additional light would be warmly welcomed.

D. C. BARD, Butte, Mont. (communication to the Secretary*).—To one familiar with the area mapped by Messrs. Billingsley and Grimes the amount of careful work involved in securing their results is apparent. The country as a whole is mountainous and not easy of access, so that many a long tramp on foot was required to accomplish the geologic mapping. The gentlemen deserve much praise for completing so well such a difficult problem, done on their own initiative and at their own expense, during week ends and vacation periods.

The theoretical conclusions of the paper are based on carefully observed field data and should in the main meet with the approval of those familiar with the geology involved.

* Received Feb. 5, 1915.

Vertical Sulphide Zones.—One statement of the author, p. 45, I desire to consider. He says, referring to the mineralization in the widely distributed veins of the Boulder batholith, "These sulphides are ranged in vertical zones within the veins; galena, with secondary silver enrichment, predominating at the surface; sphalerite most abundant between 400 and 700 ft. in depth; and below only pyrite and sparse chalcopyrite with quartz. This distribution, which conforms to the solubility of the several minerals in cooling solutions, is a universal result of the primary mineralization." I believe the above statement will not stand amplification. In the first place, I doubt the facts. I know of several prospects in the area where pyrite is practically the only sulphide mineral found in the veins, and some of these veins are high on the mountain tops. In other cases sphalerite and pyrite occur together from the surface down, galena being practically absent. Where galena does occur, I believe, with the author, that it is more plentiful in the upper parts of the vein, but I doubt if this position is due to differentiation in the primary ascending solutions. It is often true that the galena gives way in depth to sphalerite, but I hesitate to accept as a general rule the statement that the sphalerite in turn gives way to pyrite.

But even if we accept the statement of the author that the usual vertical distribution is galena, then sphalerite, then pyrite below, how can he explain the zonal arrangement by cooling of ascending solutions? It is granted that if galena is found at all it is found on top, but these galena croppings are found in the granite in valleys and on mountain tops through a vertical range of 1,500 ft. at least. It would be strange indeed if the accidents of past erosion should conform to temperature variations in the ascending solutions. Furthermore, we have to bear in mind the varying thickness of cover over the batholith at the time of vein formation and the consequent temperature differences in solutions at any given horizon.

I shall not attempt at this place to elaborate a counter theory to explain such zonal distribution of the sulphides as exists; suffice it to say that I think that it is far from a universal condition, and that where it does occur, secondary processes subsequent to the primary mineralization can explain it.

Segregations of Metals in the Magma.—In two connections in the article the author expresses his belief in local segregation of metals in the magma in excess of its average content. On p. 45 he says "it is suggestive of possible variation of the magmatic reservoirs that in the mines north and west of the general line, Butte-Wicks, lead and zinc, with the secondary silver, predominate, while to the south and east these minerals are in small amounts, with ore shoots of chalcopyrite capped by secondary chalcocite." Again, in speaking of Butte on p. 46, he says "the localization of this primary enrichment must be attributed to

a segregation of copper within the original magma, and its successive concentration by the splitting off of the several intrusions and solutions."

I should like to call attention to the fact that there need not be local concentrations of metals within the magma to produce rich ore-bodies. All that is necessary is a sufficiency of solvents capable of dissolving the normal metallic content of the magma and depositing it locally in concentrated form. Lead veins here and copper veins there need not mean that this part of the magma carried more lead and that part more copper, but rather that in one place the solutions were better adapted to dissolving lead and in the other place copper.

We often hear it said that there must have been a mighty concentration of copper in the magma beneath Anaconda Hill to yield all the copper mined from there. Now the fact is there is more copper in any cubic mile of the basalt that is found near Spokane or Pocatello than has come from the Butte district. If the same favoring solvents had occurred in the basalts as occurred in the Butte granite the Snake River plains could have had Anaconda Hills all the way from Pocatello to the Dalles. The copper is there, but there were no solutions capable of dissolving it and concentrating it into orebodies.

The condition that differentiated Anaconda Hill from Timbered Butte was not so much more copper in the magma under Anaconda Hill, but more solvents to carry it up and localize it in the veins.

PAUL BILLINGSLEY.—I am familiar with the points raised by Professor Bard and would like to state in brief just where I think they fail.

I will first take the question of the concentration of metals, as that is possibly more fresh in your minds. It seems to me here that Professor Bard's criticism is in the first place based upon a misunderstanding of the sense in which I have used the word "concentration." By this word I refer to the process by which certain portions of the original magma, left behind by the successive differentiations, become relatively predominant in the remaining reservoirs. Thus the copper, for example, originally as widely disseminated in the quartz-monzonite magma as in his Columbia River basalt, was left out of the crystallizations of the quartz-monzonite, aplite, and quartz-tourmaline periods, and was, in fact, "concentrated" into the siliceous solutions that fed the veins of the latest epoch. In the case of the Columbia River basalt the copper was caught in the basic minerals of the basalt itself. It appears to me that the instance of an acidic granite intrusion, passing through after-epochs of aplite, quartz-tourmaline, and sulphide veins, is sufficiently distinct from that of a basic lava flow to render any further comparison unnecessary.

Professor Bard's statement that varying minerals in veins of different districts are a product of the varying solubilities of their solutions to me

savors strongly of the doctrine of lateral secretion. However applicable elsewhere, this idea seems peculiarly out of place in the case of ore deposits so closely associated with the late siliceous phases of a great igneous intrusion. In the ores of the Boulder batholith I believe the evidence of a direct igneous source is overwhelming.

In the case of the vertical primary zoning of ores I believe Professor Bard has further misunderstood my position. He has stated that he doubts certain facts, but proceeds to show that these facts are not those indicated in my paper. My contention is that a vertical primary zoning—lead above, zinc next in order, and pyrite-chalcopyrite below—is a universal result of the primary mineralization of the second period. Needless to state, the expression of this zoning in the developed ore deposits is subject to the factors of distance below the batholithic cover, secondary enrichment, and varying mineral content of the solutions. I deemed it inadvisable in the limits of a paper on the broader features of the batholith to submit the details necessary to substantiate the fact of this zoning, and in fact I have an additional contribution on this subject in preparation. The following points may, however, be brought forward at present. (1) There are few mines in the Boulder batholith the veins of which outcrop at a greater depth than 1,500 ft. below the original roof. (2) With the exception of Butte, there has been no development on veins to a greater depth than 2,000 ft. below the roof. (3) The great lead-silver deposits of the batholith are in the roof itself or within 1,000 ft. of it. I refer to the Alta, Gregory, Eva May, Comet, Argenta, Bannack, and Hecla mines. (4) The chief base zinc ores are at points where the cover is more remote, as at Clancy and Butte, or on the lower levels of such mines as the Comet. (5) The few mines in the heart of the batholith, at depths of 1,500 ft. or more below the former cover, show a primary mineralization of pyrite and chalcopyrite, with none of the sphalerite and galena characteristic of the mines of the border districts. The Big Major and Bland-Weaver may be cited. (6) In no instance has secondary action removed sphalerite from the sulphide zone of a vein where it was originally present. (7) Secondary action has universally enriched the silver content for a few hundred feet below the surface, so that the upper levels of lead-rich or zinc-rich deposits have alike produced valuable silver ore.

The figures given in my paper—400 ft. depth for the lead zone, 400 to 700 for the zinc—are attempts to show their relative dimensions rather than their exact position in any or every mine.

WALDEMAR LINDGREN, Boston, Mass.—It is clearly shown, of course, that the veins are closely dependent upon the intrusion of the batholith. I may be allowed to say that I think Mr. Billingsley expressed a little too categorically his ideas regarding the arrangement of metals from top to

bottom. I think it is probably true, in a general way, but there are many exceptions to it.

There are two points I would bring out now, of the many points involved in this paper, and the first one is in regard to the relation of the silver veins of Butte to the copper veins. I never could understand it thoroughly, and would like to know Mr. Billingsley's opinion as to what relation they have to each other. As you know, they are very different. One is essentially a banded vein and the other a replacement vein without banded deposition.

The second point refers to a question in which I am deeply interested. I would ask Mr. Billingsley (who evidently leans pretty strongly toward the stoping theory, a theory which evidently has a great many strong points), whether there has or has not, in his opinion, taken place any doming of the country by reason of the pressure of the intruding rock? Personally, I am inclined to believe there has been, and am glad to see Mr. Billingsley acknowledge that in his diagram, which does show a distinct doming or uplifting of the sediments. I also wish to call attention to the fact that both Barrell, at Marysville, and Calkins, at Philipsburg, studied minor intrusions, and both have acknowledged that there is a distinct evidence of doming of batholith by the rising inclusive mass.

MR. BILLINGSLEY.—In speaking about the relation of the silver veins of Butte to the copper veins, I hope it will be understood that my opinions are not necessarily those of my associates in the work in that field, but are entirely personal.

In the section from north to south across Anaconda Hill you find the main copper veins outcropping at Anaconda Hill, the copper coming to the surface except where it is oxidized. Mines a short distance to the north reach this copper through a zone of primary lead-zinc mineralization which increases in depth with distance from the hill. In the more remote veins there are three zones, rhodonite, quartz, and silver occupying the veins above the zinc, and in some of the northern mines reaching a depth of several hundred feet before the zinc is reached. A section across the camp from west to east on any of the main veins shows the same thing. Successive levels of the mines extend beyond the copper into a mineralization of the lead-zinc type, while outlying shafts show above this the additional rhodonite-silver zone.

The copper zone extends its boundaries with increasing depth, but is apparently unconformable with the silver and zinc zones, which pass below into lean quartz, pyrite, chalcopyrite ore. In general the rich copper sulphides—enargite, bornite, covellite, and chalcocite—have merely enriched the lean quartz-pyrite of this lower zone, but at numerous places they have penetrated into the zinc ores. In such instances the details are irregular, but the copper distinctly is a later

addition to a pre-existing mineralization. Beyond question a certain amount of replacement of zinc by copper has occurred, and some migration of the replaced zinc (an idea originated by Murl Gidel of Butte) is suggested by the presence of impure sphalerite in many of the recent fissures.

The earlier mineralization—silver, zinc, and chalcopyrite—is similar in type to that of the main veins of the other granite districts, and I am inclined to regard it as of the same origin. On this hypothesis the early Butte ores are in genesis and type no different from those formed over widespread areas at a certain stage in the evolution of the batholith. The primary enrichment in copper, however, is a special local feature, occurring at a later point of time and requiring special factors, which are unknown, to explain its presence.

The question of doming may best be approached by a cross-section of the batholith. The granite appears as a dome, with double crests north and south of Basin separated by a deep trough. Above the dome is andesite, which was deposited as flows upon the surface of tilted sedimentary rocks. This surface, which is older than the granite, makes a satisfactory datum plane for determining possible deformation caused by the intrusion. Fragments of this now remain wherever, in the bits of cover left, we find andesite resting upon sedimentary rocks. Above the high points of the dome this old surface has now an elevation of over 8,000 ft. At Garrison, 40 miles west, its altitude is about 5,000 ft. At Radersburg, 50 miles to the southeast, it is less than 4,500. It therefore appears that over the high points of the granite intrusion a pre-existing surface is now found at elevations of 3,000 ft. greater than on the flanks of the igneous mass. This may be a coincidence, but it certainly suggests some uplift of the country. There is, however, almost no evidence of dynamic disturbance of the sedimentary rocks at the actual contacts.

HORACE V. WINCHELL, Minneapolis, Minn.—One of the greatest values of papers of this sort is the bringing together of facts which are probably related to each other, but which, if observed singly and reported upon separately, do not appear to have the bearing which they do actually possess. The conclusions here drawn, based upon such a large number of widely separated observations, are, therefore, the more to be relied upon.

I have wondered in listening to Mr. Billingsley whether it was possible to attribute all of the mineralization of Butte to agencies arising within the one mass of rock, the Boulder batholith, and have noticed the absence of any reference to the quartz porphyry, with which the copper veins of Butte are characteristically associated. The large veins are closely related, geographically, in their strike and dip and general

position with quartz-porphyry dikes. In his paper, I believe there is some reference to a differentiation of these dikes, changes of them in depth, but it seems to one who has studied the camp in former years that there must be some genetic relation between the intrusion of quartz porphyry in the monzonite or the batholith itself, and the ores, and that there may possibly be also some relation between the rhyolite and andesite and the silver mineralization; and it requires a rather extended imagination to conceive of the entire mineralization of Butte in the different kinds of ores, the different kinds of veins, both typically banded quartz-manganese veins and the large and extended replacement copper veins, as all to be assigned to one large period of mineralization.

The idea which I think Mr. Billingsley has of the zonal arrangement perhaps leaves us with the impression that every mine in Butte, especially the large mines upon the borders, may present illustrations. His sketch upon the blackboard during his last remarks gave me the idea that when he said that there were 800 or 900 ft. of quartz-manganese and silver ore on the surface, and 1,500 ft. of zinc beneath, it had been demonstrated that beneath this aggregate of 2,400 ft. copper had been found. That does not represent my understanding of the facts. The large mines in which the zinc has been found have not yet turned into copper mines in the bottom, although some copper ore is found in veins which cross the zinc veins, and may belong to a different period.

MR. BILLINGSLEY.—The last point brought up by Mr. Winchell may, perhaps, be answered by naming the Badger State and the Elm Orlu as two mines in which the conditions which I have stated do exist. I did not, however, intend to convey the impression of the general presence of copper in depth. I have purposely refrained from going into definite details in the case of the Butte district.

In regard to the influence of rhyolites on ore deposits, it must be remembered that so far as we know the rhyolite is later than any of the mineralization. In the Hornet mine, near Butte, the rhyolite cuts and displaces one of the main veins of the district. The same is true throughout the batholith. In the Comet mine, near Basin, a rhyolite dike displaces the vein about 50 ft.

I have also been rather chary of mentioning the quartz porphyry, because in the lower levels of the Butte mines, where we should find the lower extension of the quartz-porphyry dikes, we find frequently masses of aplite. The point is unsettled whether the quartz porphyry can be separated from the aplite in many of these occurrences.

In one case Mr. Sales called my attention to a crosscut containing quartz porphyry, while another crosscut 25 ft. away contained aplite. In view of this fact, we do not say much about the quartz porphyry until we find out whether or not it is distinct from the aplite. If it can be

proved that it is distinct, it will satisfactorily explain to all of us the latest mineralization in Butte.

L. C. GRATON, Boston, Mass.—This question of the distribution of the metals in vertical zones is assuredly one of utmost interest. If Professor Bard intends his conclusions to apply to the zoning at Butte, the only camp in the region described of which I may speak from personal observation, I am inclined to take issue with him and agree with Mr. Billingsley that the zonal disposition of the metals is rather a circumstance of original deposition than a result of later rearrangement by superficial agencies. On the other hand, it seems to me that so far as Butte is concerned there may be another explanation, or possibly a modification of the explanation that Mr. Billingsley has given, for this distinctly evident change in the character of metallic content outward from what may be regarded as the center of mineralization.

In his discussion of this subject I believe Mr. Billingsley has several times used the word "replacement." If by replacement he means simply that one metal may be found at one part of a vein or of the district, while another metal predominates in another part of the vein or the district, no exception can be taken; but if he means that zinc, for instance, was originally present at the spot where copper now is, and that the zinc has actually been driven out by the copper—that is, replaced in the strict sense—I do not feel satisfied by those microscopic studies with which I am familiar that such is the chief explanation of this certainly marked and very interesting zonal arrangement of metals which Mr. Billingsley has described and pictured.

Mr. Winchell has pointed out what seemed possibly to be some idealization of this matter of zoning, to which I have no doubt Mr. Billingsley resorted simply to make the situation perfectly clear. Though my understanding is, of course, not at all comparable with Mr. Billingsley's long and intimate knowledge of the camp, I think Mr. Billingsley would probably have us understand that sharp lines separating zinc and copper, such as he has drawn on the board, actually do not exist, but instead there is a gradual transition, with here and there some overlapping, as indicated by the tongues of copper-bearing ground which in his sketch are seen penetrating into the zinc-bearing region.

This particular subject of zoning is one worthy of much study, to determine whether one metal or mineral actually does come along and to an important degree drive another one out, leaving, perhaps, actual physical remains of the first in greater and greater proportion with increasing distance from the central source, or whether, on the other hand, a progressive change in temperature and chemical equilibrium causes a precipitation of different metals or minerals in gradually changing proportions as part of a single process of deposition. Actual replacement

of one set of minerals by another is probably better illustrated at Butte than in most districts. For my own part, however, I am not altogether convinced that such replacement has been the controlling influence in the large-scale concentric segregation of the metals illustrated in this camp.

The detailed paper by Reno Sales upon the ore deposits of the Butte district proper, presented a year and a half ago, excellent as it was, I feel gains, is put in a truer perspective, by this paper of Mr. Billingsley's, which deals with the surrounding region and furnishes, so to speak, the setting of the gem. A very high standard of geological work has been set by these gentlemen in their treatment of the Boulder batholith and its principal locus of ore deposition. They are surely to be congratulated upon this splendid showing, and the Anaconda company commended for the encouragement and support it has afforded through them to the practical application of scientific geology.

Method for the Determination of Gold and Silver in Cyanide Solutions

Discussion of the paper of L. W. BAHNEY, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 339 to 344.

E. J. HALL,* New York, N. Y. (communication to the Secretary†).—The Chiddy method for determining gold and silver in cyanide solutions has been subjected to so many proposed modifications, of which L. W. Bahney's is one, it is natural to suppose the method is defective.

There is no denying that this method is capable of giving inaccurate results, but so is any other method of quantitative determination. However, proof has not come to my notice that the troubles are inherent and not in the detail application, and I am inclined to think from our experience that in many cases it is the latter.

Errors in this method may be occasioned by:

(1) The retention of zinc in the sponge, which tends to desilverize the molten lead in cupellation, similar to the action in the Parkes process, removing gold and silver in a crust of zinc oxide. Failure to remove zinc may result from lack of concentration of hydrochloric acid, due to neutralization by salts present, volume of liquid too great for the specified amount of acid, or the use of weak acid. (HCl rapidly loses its strength unless kept in perfectly tight containers. The HCl gas escapes even from the ordinary glass-stoppered acid bottles after the plaster seal has been broken.) Undissolved zinc is likely to be held in the upper part of the lead sponge, which floats above the liquid, and unless the solution is boiled it will escape action. Mr. Bahney's modification makes this

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† Received Mar. 28, 1915.

difficulty impossible. It is not, however, a necessary evil when the assay is properly conducted.

(2) In the assay of foul solutions the sponge is apt to disintegrate, as pointed out. Occasionally this tendency may be overcome by slightly oxidizing the solution after adding zinc and boiling to reduce impurities. The lead acetate and remaining acid is then added. When the sponge breaks up it is usually sufficient to decant most of the liquid and then wash the lead into a boat, drain off remaining water, and cupel. If the lead is fine enough to remain in suspension, prohibiting decantation, it may be rapidly filtered on SS No. 597 filter paper. Fluxing is not necessary, as the paper may be pressed between blotters, sponge separated, paper folded once, held by one end with a pair of tongs and the bottom ignited with a bunsen burner, allowing ashes to fall into a lead boat containing the sponge; then wrapped and cupelled.

(3) The furnace temperature necessary to start cupellation is considerably higher than that required for a good finish. The major portion of cupellation loss occurs when the last gram or two of lead is being oxidized. This loss increases rapidly with increase in temperature, particularly with small beads such as are usually obtained from cyanide solutions.

There is about 1.7 g. of lead in 10 cc. of a saturated lead acetate solution, the quantity usually employed. The maximum weight of sponge will not exceed this weight and is often less than 1 g. when the solution has been heated for a long time or HCl concentration is high. The sheet lead required for wrapping is 2 to 3 g., so the whole packet will not weigh over 3 to 5 g.

For the reasons given above it is impossible to obtain good results in cupelling lead buttons of this size unless the cupels are moved to the mouth of muffle or coolers introduced in the furnace immediately after the buttons have opened. This is particularly true when large cupels are used, as they give up their heat slowly. If gold is the principal metal sought it should always be protected by silver.

That gold and silver in all cyanide solutions can be best determined by any one method is not a reasonable assumption. For rich solutions where 1 to 2 assay tons will suffice and prompt returns are not required the evaporation method in a lead boat is most attractive, as the working time and attention are least. A lead boat 1.75 by 3.75 in. made from a sheet 3 by 5 in. will permit evaporation of 2 assay tons of solution in 60 to 75 min. A covering of test lead on the bottom of boat will facilitate evaporation and reduce the tendency to spit. If the solution is foul or carries suspended matter, scorification is desirable.

Mr. Bahney objects to this method because the lead foil may leak. Assayers' sheet lead holds water better than the foil and is readily obtainable.

Solutions that are reasonably free from impurities and suspended matter give good results by the Chiddy method and its modifications. Suspended matter not soluble in HCl will collect in the sponge, as well as other impurities, and scorification should follow. Under these conditions one of the so-called precipitation methods is preferable.

It is unfortunate that Mr. Bahney did not include some comparative figures, particularly as the assumption that some gold and silver might be retained on the zinc stick seems reasonable.

The Mining and Reduction of Quicksilver Ore at the Oceanic Mine, Cambria, Cal.

Discussion of the paper of C. A. HEBERLEIN, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 497 to 504.

H. D. PALLISTER, State College, Pa. (communication to the Secretary).—Mr. Heberlein gives some valuable information concerning a branch of mining and metallurgy which has been neglected during the last few years. His results with fine grinding and Deister tables suggest the possibility of the mining and concentration of a number of low-grade cinnabar deposits. Can Mr. Heberlein give us some idea as to the loss of quicksilver during this concentration?

The advances in the metallurgy of quicksilver have been few and far between in recent years. Most miners prefer to go on in the old way with the Scott furnace and the large brick condensers. The large brick condensers are expensive to build and to keep in repair, and on account of the thick walls are extremely slow in cooling the gases; and further, the joints between the bricks offer passages for the quicksilver vapors unless great care is used in building the condensers. The escape of vapor at the top of the furnace is another source of loss and I see the people at the Oceanic mine have overcome this in a very ingenious way.

I recently returned from the Terlingua quicksilver district of Texas, where the only operating company, the Chisos Mining Co., is operating a 20-ton furnace of the Scott type. They had considerable difficulty with getting a good extraction, due to too high heating in the furnace with its resulting difficulties. Recently they have been running with ashpit and fire doors closed and have been able to keep the heat lower, thus getting much better results and practically verifying the results obtained at the Oceanic mine.

By keeping down the quantity of air the heat is reduced below a point where the pyrite completely dissociates; hence, less sulphuric and sulphurous acid is formed and therefore less of the quicksilver is attacked and converted into compounds which are more difficult to recover. Another effect of a larger quantity of air is the additional

bulk of gas and vapor that must be handled, requiring larger condensing apparatus and greater care in the condensation, which is usually more difficult with the larger bulk of vapor. Has Mr. Heberlein ever attempted to determine the loss of quicksilver in the gas escaping from the stack, and if so what has been the result? Also at the Oceanic plant where are the dampers placed and how kept and is artificial or only natural draft made use of in running the furnace and condenser system?

Mr. Heberlein speaks of assaying and panning and I am interested in knowing what method of assaying he uses and if he finds any trouble in getting correct results. In the Terlingua quicksilver district the hornspoon is used to test the material in the faces and is accurate in the hands of an experienced man up to about 1 per cent. The hornspoon is also of value in determining the condition of the quicksilver, that is whether it is there in the form of cinnabar, native, or some of the rarer mineral forms. In the accurate work at Terlingua the Whittom apparatus and method are used for assaying the quicksilver ores and cinder. This method consists briefly in mixing a gram of ore with its own weight of iron filings (free from grease), placing in a special steel crucible, and covering with more iron filings. Next a plate of silver foil covers the crucible and is held in place by a water container, ring and clamp. This apparatus is then placed over the flame of a Dangler gasoline burner, the water cooler being filled with cold water, and heated for about 20 min. The apparatus is then removed from the flame, the clamp unfastened and the silver plate weighed. The increase in weight is quicksilver, and the percentage can be easily calculated. My experience has been that this method is accurate unless the ore contains bituminous matter or pyrite, when unless great care is used the results will be misleading. By treating carefully with alcohol, ether, or chloroform, the bituminous matter can usually be greatly reduced if not entirely removed, but if heated too hot or too long, any sulphur formed is impossible to remove with the chemicals tried. Can any one suggest a remedy?

In getting an extraction of 90 per cent. Mr. Heberlein is doing very well for quicksilver extraction. Can he account for the other 10 per cent.? It has been my experience that the losses which occur are hard to figure as well as determine where they are escaping. To get the percentage of extraction and hence the loss, it is usually necessary to take the complete run from the time the furnace is started until it is closed down and cleaned up, because during the first month of the run most of the quicksilver goes to the coating of the inside of the condensers and pipes and is not recovered until the cleanup. After this extraction has been added to the other losses that can be accounted for, such as the amount in the cinder, that in the water, etc., there usually remains a considerable amount which is unaccounted for and must be lost. The question is, where does it go? Some of it may go out the top of the

furnace, out the stack, through the joints of the condensers and furnace, and in some of the older furnaces out through the bottoms.

It seems to me that the ideal furnace for the treatment of quick-silver ores should be one in which the flow of air over the ore can be under control at all times and one in which the vapors can be kept from coming in contact with the gases from the source of heat. In other words, a continuous muffle furnace.

Metallurgical Practice in the Porcupine District

Discussion of the paper of NOEL CUNNINGHAM, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 99, March, 1915, pp. 601 to 608.

C. H. POIRIER, New York, N. Y. (communication to the Secretary*).
—Mr. Cunningham's paper, written from the standpoint of a stamp advocate, while commenting more or less favorably upon the installation of ball mills in the Vipond plant, does not, in my opinion, give as much emphasis to the comparatively new departure as the installation of ball mills in the place of stamps justifies. This discussion is based upon my personal and intimate practical work for several years with stamp mills, and for the past three years with Hardinge ball and pebble mills. The fact that the Vipond mine contains about equal quantities of the A and B classes of ore mentioned by Mr. Cunningham, and the further fact that I have had a most intimate knowledge of the working of both classes of ore in the Porcupine district in stamp mills and ball mills, justifies upholding my final decision to use ball mills in the place of stamps, and defends my present further action in now increasing the present Vipond capacity by the addition of another 6-ft. Hardinge ball mill, in which action the directors of my company concur. This is one of the strongest evidences I can possibly offer in substantiation of my opinion that the ball mill exceeds the efficiency of the stamp, both in metallurgical and mechanical economy.

It is hardly fair to compare a 100-ton per day plant with a mill handling several times this amount, yet it is my belief that the cost per ton of grinding in the ball mill is far less than that of any stamp mill in the district, where capacities run up to 400 tons per day.

Mr. Cunningham also states that in his opinion the long cylindrical tube mill should be used, if only for theoretical reasons. For practical economic reasons I am employing in the Vipond plant fine slimming pebble mills of the Hardinge type. These are only 6 ft. in cylindrical length, are operated in a closed circuit, and are doing practically the same amount of work that tube mills two or three times their

* Received Mar. 27, 1915.

length are doing in other plants of the district, and at a much less cost for wear, tear, and power.

Three years ago our present total 100-ton crushing equipment, including foundations, building, and power machinery, was installed for approximately \$27,000. The upkeep and attendance of a ball mill with an approximate consumption of $\frac{1}{2}$ lb. of balls and from $\frac{1}{20}$ to $\frac{1}{10}$ lb. of lining per ton of ore ground, including power, wear and tear, etc., does not exceed 10c. per ton, when taking the product of the rock crusher passing a 2-in. ring and reducing it so that all practically passes an 8-mesh screen. This product is sent directly to the Hardinge pebble mills after classifying out about 25 per cent. of minus 200-mesh material. A closed circuit is maintained on the pebble mill. The cost of sliming, including pebbles, power, wear and tear, etc., in the pebble mill approximates 10c. per ton, or a total of approximately 20c. per ton for grinding from rock crusher to cyanide tanks.

Herewith I submit a grading analysis of ball-mill feed and product. The feed to ball mill is the product of a rock crusher, all passing 2-in. mesh:

Mesh	Feed to Mill, Per Cent.	Product from Mill, Per Cent.
+ 1	5.50	
+ $\frac{3}{4}$	28.00	
+ $\frac{1}{2}$	30.00	
+ $\frac{3}{4}$	19.72	
+ 10	10.87	2.10
+ 20	2.42	8.00
- 20	3.51	
+ 40		22.68
+ 60		10.50
+ 80		11.80
+100		3.90
-100		40.15

I am under the impression that our equipment is the simplest and most up-to-date in existence to-day, and, as I fathered ball-mill crushing in opposition to stamps in the Porcupine district, I am inclined now to support strongly my original opinion that it is a pronounced success; hence this discussion of Mr. Cunningham's paper relative to the Vipond.

Now, referring to the mention of the McIntyre installation—the McIntyre originally had a stamp equipment, which was discarded for a Chile mill, and finally, based upon the work done in our Vipond mills, one 6-ft. ball mill was installed in the McIntyre plant, and later, when an increase in capacity was necessitated, another Hardinge ball mill was added.

Recent Developments in Coal Briquetting

Discussion of the paper of CHARLES T. MALCOLMSON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 271 to 289.

E. W. PARKER, Washington, D. C.—The briquet or boulet, or coalette (whatever name is used) has come to stay as a part of the fuel supply, particularly for domestic purposes. During the past winter I have had some personal experience with the use of boulets made from anthracite coal. All of this winter I have used boulets in my furnace and in my grate at my home in Washington. One reason for doing so is that we get the boulets at a price considerably cheaper than the prepared sizes of anthracite coal. The fuel has proved entirely satisfactory. One advantage possessed by the boulet is its absolutely uniform size, which makes the ventilation of the firebed practically perfect. The boulets burn until entirely consumed. There is not, even in an open grate fire, if the fire is kept constantly going, any unconsumed fuel passing through the grate bars. The boulets hold their form until entirely consumed and then break away into a clean white ash, with absolutely no clinker. That in itself is a particularly favorable recommendation to the housekeeper.

Mr. Malcolmson has mentioned the fact that notwithstanding the smokeless character of anthracite, the boulets made from it emit smoke. The emission of smoke is due to the coal-tar pitch used as a binder. This is particularly noticeable if the fireman is somewhat careless, or wants to save labor, and feeds too much of the fuel on the fire at one time, cooling it down and thus causing an unnecessary amount of smoke. The smoke lasts but a short time, however, and if the fire is fed a little more frequently, with a small quantity of fuel at one time, the smoke is reduced to a minimum. I believe it will not be long before some of our engineers will have overcome the objectionable smoke, and will produce either from anthracite or from the so-called smokeless New River and Pocahontas semi-bituminous coals, a briquet which will be as smokeless as anthracite for all practical purposes.

Mr. Malcolmson in speaking of some of the financial wrecks that have marked the progress of the development of the briquetting industry has omitted to mention one factor which, I think, has had more influence in this respect than any other. This has been the endeavor to exploit secret or patented processes and binders which, when shown to the over-anxious prospective investor, showed very remarkable results as a fuel. The briquets burned well and appeared to carry out the claims made by the promoters, but when put upon a practical basis for commercial operation the projects have resulted in failure.

There is, on the Pacific Coast, an interesting development in the manufacture of briquets not made from coal at all. The ordinary manufactured fuel and illuminating gas in Los Angeles, Cal., and Portland, Ore., is made from petroleum. The residue consists of material which is called carbon, as it really is. At Los Angeles for a number of years the Los Angeles Gas & Electrical Corporation has been briquetting this residue, and making a very acceptable domestic fuel in briquetted form. A former waste product has been converted into an excellent fuel. Within the last year a similar plant was erected at Portland.

ALFRED C. LANE, Tufts College, Mass.—Is there any probability or any possibility that the method of using coal dust directly, in producing gas, or as a fuel, might use up all the coal dust, without briquetting?

E. W. PARKER.—I hardly think so. Some powdered fuel is used in cement manufacture, in which case the ash in the coal goes into the clinker and really adds to the output of cement. For boiler practice, however, the use of powdered fuel has not, so far as I know, proved successful. The intense heat created necessarily fuses the ash, and it is deposited as obsidian on the bridge walls, boiler tubes, etc. It makes as perfect an insulation as one could possibly desire.

A. C. LANE.—I had reference also to the system developed by Pintsch and Diesel, where the anthracite coal is burned and the fuel gases are recovered and used in a reservoir.

E. W. PARKER.—I do not know of any really successful use of powdered fuel in the manufacture of producer gas or other gases. Some of the smaller sizes of coal are used for that purpose, but at Point Breeze, at the lower part of Philadelphia, the United Gas Improvement Co. for a number of years has briquetted fine coal, both anthracite and bituminous, with a mixture of coke breeze. Water gas was made from the briquetted fuel.

CHARLES DORRANCE, JR., Lansford, Pa.—In regard to the use of powdered anthracite as a fuel, I would say that for Mr. Lathrop, of the Lehigh Coal & Navigation Co., a number of years ago, I ran a test on a Barnhurst furnace burning pulverized anthracite coal. The coal, anthracite culm or slush, was pulverized in a cement mill, through 80 mesh, and was then blown through a tuyère into a specially constructed combustion chamber, so designed as to give a radial motion to the flame and fuel as it went in. The combustion chamber was heated first with bituminous coal, and after the temperature of the combustion chamber was high enough, the anthracite coal was introduced instead of the bituminous, and a very intense blast-furnace-like heat was maintained. We ran two 24-hr. boiler tests on that and developed some hours as high as 50 per cent. over the rating of the boiler. The main trouble in the

whole process seemed to be that the refractory which formed the lining of the combustion chamber would not stand the blowpipe action of the flame, and the scouring action there would eat the lining out in a few weeks' time.

The other chief disadvantage of the process was pulverizing the anthracite coal. Pulverizing bituminous coal is simple, and is done all over the cement region to-day, but the anthracite coal, having a cubical fracture, does not pulverize to dust, but goes down on to its different cleavage planes smaller and smaller, and we figured out that to pulverize anthracite successfully on a commercial basis would cost in the neighborhood of 20c. to 30c. per ton. When you figure that the price of No. 3 buck or barley anthracite coal now is anywhere from 30c. to 50c. a ton, the feasibility of pulverizing anthracite coal as a fuel is doubtful. I have also the statement of a mechanical engineer of the biggest cement plant in the Lehigh region, in fact in America, that they have spent a great deal of money in trying to use powdered anthracite coal in their cement industry, and have given it up as being very much more expensive than either run-of-mine bituminous or pulverized bituminous coal would be.

EDWIN LUDLOW, Lansford, Pa.—In regard to the briquetting of anthracite coal, the Lehigh Coal & Navigation Co. built a briquetting plant several years ago, and has been manufacturing briquets ever since, and produced the boulets or briquets that Mr. Parker has just spoken of.

The briquets are made of 93 per cent. of anthracite culm and 7 per cent. of dry pitch binder; the materials are heated by steam, and thoroughly intermixed in a pug mill until the mixture has what we call a "blue mix color;" it is then run through a press, making boulets of a size intermediate between nut and stove coal, especially adapted for domestic purposes.

Although we sell these boulets at \$1.50 to \$2 a ton less than nut coal, we have found no very large demand for them; in fact, it is difficult to get repeat orders on account of the smoke in the binder. This smoke is not troublesome to the man using the boulets, as by keeping the draft on for not more than 5 min., when the fire first starts, the smoke is burned off and the boulets then make a bright, clear fire superior to anthracite for an open grate or for a stove. But this smoke is very heavy, with a pungent odor, and after passing out of the chimney of the man who is burning the boulets it descends into the windows of the man next door, and the objection to the use of boulets comes from the neighbor, rather than the man who is burning them.

In order to overcome this objection, we have been manufacturing boulets with a smokeless binder that gives equally good burning results

and absolutely no smoke; the objection to these boulets is that they are not waterproof, and will soften if exposed to the weather. Mr. Malcolmson is carrying on experiments with the weatherproofing of this class of boulets and assures us that it is entirely practicable, and we propose trying out these experiments on a commercial scale to test the laboratory results that Mr. Malcolmson has obtained.

It is perfectly true, as Mr. Malcolmson states, that the anthracite companies as a rule are not giving the question of briquetting anthracite culm very serious attention at this time; the last two years having been years of slack demand, so that the anthracite companies have not been able to run full time and all of them have accumulated large stocks of prepared sizes. It would not be policy for them to decrease the sales of their own prepared sizes by manufacturing a fuel from a waste product to compete with them.

The anthracite companies, however, are prepared for the day that we all look forward to, when the demand for the prepared sizes of anthracite will exceed the production of the collieries, and when that day comes, the companies will be found with large stocks of culm ready to be made into briquets as fast as the trade will take them.

Formerly, anthracite culm was thrown on the waste banks with the breaker slate, but at the present time most of the companies are separating their culm, and where it is not being used for slushing operations underground, it is being stored in preparation for the day when there will be a market for it; but until the market will absorb the prepared sizes it is obviously better for the companies to store their waste product of slush rather than their prepared sizes, that represent an investment of from \$3 to \$4 a ton.

W. H. BLAUVELT, Syracuse, N. Y. (communication to the Secretary*).—Mr. Malcolmson's paper is of special value to those who are interested in the briquetting industry either on account of its important relation to the conservation of the fuel supply of our country or for more personal reasons; because it describes several plants which have been established on a sound commercial basis and give every indication of permanent and successful operation. Unfortunately, the record of the undertakings for the briquetting of coal in this country has been heretofore practically one of unsuccessful attempts. Failures have been due in most cases to lack of appreciation of all that goes to make a successful briquetting operation. In several cases only one or two seemingly minor operating details prevented success. Other installations failed on account of lack of suitable market. Either there was not enough difference between the price of the raw material and the finished product, or there was no place for the new fuel in competition with other fuels already in use. It is satisfactory to note that in the plants described by the author of the

* Received Feb. 15, 1915.

paper all the conditions seem to have been met and the plants apparently have come to stay. With these installations to open the way, it is reasonable to expect that other plants will follow very soon.

I am especially interested in these satisfactory installations on account of my own experience with a briquetting plant, which I reported to the Institute in a paper read at the Pittsburgh meeting in 1910. This paper described a plant built by the Solvay companies at Detroit primarily for the manufacture of briquets from coke breeze from their coke ovens. The plant described in my paper was worked out to a commercial success, so far as the cost of manufacture and quality of product were concerned, and a very satisfactory product was marketed for some time at a profit. After my paper was presented some experiments were carried out in the baking of the briquets in order to make them smokeless. These experiments were entirely successful. Sufficient coal was mixed with the coke breeze so that with the pitch binder there was enough cementing material to make a strong briquet, which produced no smoke in burning, retained its original form, and yet was coherent enough not to break down in handling. But after the briquets began to be thoroughly established in the open market it was found that they were competing directly with another product of the company, namely, domestic coke, so that every ton of briquets that was placed meant the sale of one less ton of domestic coke. As the result of this entirely unforeseen condition the briquetting plant has not been in operation for several years. This experience is an excellent illustration of the careful study that must be given to all the conditions before undertaking the manufacture of briquets.

The author's description of the successful use as a binder for briquets of the asphalt obtained from the distillation of petroleum oils of asphalt base is of special interest, particularly in connection with briquetting plants on the Pacific Coast. Coal-tar pitch is the only other binder that has been generally successful either in this country or Europe, as practically all the other binders either fail to make a waterproof briquet, or else they add materially to the percentage of incombustible matter in the finished product. It would appear that America cannot much longer postpone the utilization of coal tar in the manufacture of the multitude of useful products of which it is the base. When these industries have achieved the important position in America which they deserve, and which they could occupy under the protection of the intelligent legislation which we hope will grow out of the lessons we have been learning during the past six months, let us hope that the briquetting industry will also have developed to a point where it will offer a sufficient market for the pitch which is the residue from the first step in the manufacture of the coal-tar products.

Depreciation as Applied to Oil Properties

Discussion of the paper of PHILIP W. HENRY, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 97, January, 1915, pp. 23 to 30.

C. E. GRUNSKY, JR., San Francisco, Cal. (communication to the Secretary*).—The difference of opinion of engineers on the subject of depreciation is justified, to some extent, by the many viewpoints from which such a subject can be considered when applied to properties of widely differing physical characteristics.

Mr. Henry takes up the subject of depreciation as applied alone to oil properties, and in doing so makes a clear distinction between amortization and the repair or renewal requirement.

After carefully reading the paper the purpose of its author in suggesting a "depreciation" reserve or fund to keep the property at a *constant value* does not seem clear. As shown by Mr. Henry, the depreciation element as compared with the operating expense per gallon of oil is very large because of the comparatively short life of the oil properties, and it is also evident that such a "depreciation" fund would in a short time contain a very large amount of money.

It seems desirable and permissible in income-tax matters to take into account the depreciation of property and to separate from the income such money as is necessary to amortize the invested capital to an extent warranted by the depreciation. However, it does not seem either necessary or desirable to create a large fund which will in time approximate the entire invested capital and which may be invested by the directors, as Mr. Henry suggests, "in the property itself or in *outside securities*."

This method seems cumbersome and liable to offer in the reinvestment of such a fund temptations to a directorate that might show more devotion to its own interests than that of the stockholders.

A reserve fund is very frequently desirable in mining and oil operations, but to keep up a fund of this magnitude or to reinvest it in outside securities does not seem desirable from a stockholders' standpoint. The money withdrawn from the receipts to be placed in the depreciation fund (not including replacements and the drilling of new wells necessary to keep up the productiveness) should be returned to the stockholders of the company, not as dividends, but as an amortization of the invested capital.

Past practice has been to return the invested capital as dividends, no differentiation being made between profits and returned capital, and no provision being made for any fund called a "depreciation" fund, or what you like, for the purpose of amortizing the capital. During the life of a property (annually if possible) all of this depreciation fund should be returned to the stockholders, excepting such portion as it is desirable to

* Received Feb. 1, 1915.

retain intact to insure the continuity of operations, and in such event outside investment from this fund by the directors should be prohibited.

In Form No. 1031 (revised), *Return of Net Annual Income—Corporations*, issued by the Internal Revenue Bureau, under the heading "5 (B) Depreciation and Depletion," the following limitation is placed on the amount to be deducted from income because of the exhaustion or the depletion of natural deposits: "In computing depletion in the case of natural deposits the rate should not exceed 5 per cent. of the gross value at the mine of the output for the year."

This requirement determines that the total amount deducted from the gross income during the life of the property, which is to be laid aside or returned to the investor as capital, shall not be more than $\frac{1}{20}$ of the gross value of the total production from the mine or oil property. If, under this ruling, the owners or investors pay a large sum of money for the property or erect an expensive plant because they can recover the product at a low cost per ton or per barrel and the investment exceeds this $\frac{1}{20}$ of the gross value of the total production, a portion only of their capital will be returned as dividends with the income tax subtracted from it.

The proper way to estimate this "depletion" is, as pointed out by Mr. Henry, to use as a basis for the estimated life the average life of the oil properties in the vicinity of any property. Then the annual depletion will be the annual installment which is necessary to return to the owner his legitimate or proper investment during that period of years.

When it is observed that the actual life of an oil property will be prolonged beyond that originally expected, the estimated life can be increased, and when it is seen that the output is decreasing rapidly, the reverse is possible. Under no circumstances, of course, should the owner be allowed to credit, because of the depletion of his property, more money to the depreciation fund than has been legitimately invested by him.

The fact that the remaining life of the properties may be extended from time to time is discussed at some length by C. E. Grunsky in a paper published in the *Transactions of the American Society of Civil Engineers*.¹ Those who care to go more deeply into this phase of the question may refer to this paper.

I. N. KNAPP, Ardmore, Pa. (communication to the Secretary*).—As a general accounting proposition on depreciation, Mr. Henry's paper is undoubtedly correct. In discussing his paper I can only give my

* Received Mar. 2, 1915.

¹ Depreciation as an Element for Consideration in the Appraisal of Public Service Properties, *Transactions of the American Society of Civil Engineers*, vol. xl, No. 9 (Nov., 1914).

personal experience and opinion from the standpoint of an oil and gas operator.

The depreciation on buildings, machinery, tanks, and pipe lines may be readily agreed upon, as they are on the surface and are easily examined as to their condition from time to time, and a civil engineer used to such public utilities would be entirely competent to pass on that part of the problem.

A government ruling as to depreciation or losses is too often a "heads I win, tails you lose" proposition which may have to be referred to the courts to tell what is meant. Such rulings may be applicable to the collection of taxes, but do not really tend to show whether any particular property or business is paying or not, or what may be the real depreciation.

To be properly qualified to estimate on the depreciation of oil and gas producing wells, which necessarily reach many feet below the surface, requires knowledge that can be acquired only by actual experience in drilling and operating.

The general public is very much misinformed as to the actual costs and depreciations of oil and gas properties. The fakir, the boomer of fraudulent oil and gas promotions, the space writer for the magazines, all have spread abroad a sort of idea that all that was necessary to get a big-paying property was to hire a driller to put a few pipes in the ground most anywhere and tap vast stores of oil and gas provided by nature and get a perennial yield of oil and gas at a slight cost and no depreciation.

These ideas are reflected in legislation and decisions of the courts.

To assist the oil and gas producer to meet unwise legislation and imposition of unreasonable taxation and bureaucratic interferences, the producer must inform himself as to what the itemized costs of producing are and what the depreciation is on his properties.

Some years ago I operated in a small oil pool and kept my accounts according to a classification outlined by an accomplished accountant.

A record of all bills paid and the items received was sent in for monthly audit and a trial balance returned, thus giving a monthly check on the business.

Once a year a complete inventory was made, using the same measure of value for each producing well, and the depreciation for the year worked out. The routine work was done by clerks, but the trial balances and inventory sheets were checked and signed by the accountant.

We found the depreciation in yield on about 60 oil wells regularly operated was 3 per cent. per month, or about 28 per cent. per year. The total cost (including all proper charges and depreciation) of putting the oil into the field tanks was 53c. per barrel for a period of years.

The yield from gas wells was so erratic and measurements so uncertain that I could never get a satisfactory line on the cost of production or de-

preciation. I regret that the books of this operation are no longer in existence, as they gave costs and depreciation in detail.

The wells were pulled out when from five to seven years old because oil fell to 27c. per barrel; it had been as high as \$1.38.

The operation had already paid out and made considerable profit, but our books showed that there was no margin at 27c. for continued operation.

Possibly it would have paid to have kept the property for a rise in the price of oil. The reason for not doing so was the probable excessive depreciation of the property and wells when not regularly operated, the heavy taxation put on oil and gas properties; also the majority of the land owners insisted on continuous operation.

The Use of Mud-Laden Water in Drilling Wells

Discussion of the paper of I. N. KNAPP, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 96, December, 1914, pp. 2783 to 2793.

A. C. LANE, Tufts College, Mass.—Is there any noticeable effect in drilling against salt water? Mr. Knapp says that the mud settles very quickly when brine is added. What is the effect when you strike a strong salt water in boring?

I. N. KNAPP.—You are supposed to have enough mud pressure in your well to keep the salt water out. I have drilled against what I knew to be 80 ft. head of water, and hot water too at 99.5°; this water did not show until the well was bailed and the mud pressure reduced.

HENRY LOUIS, Newcastle-upon-Tyne, England (communication to the Secretary*).—Mr. Knapp may be interested to learn that the use of mud in drilling through porous strata is much older than he appears to be aware of. It has been used notably by Honigmann for boring shafts in Holland and in some of the adjoining portions of Germany. At the Oranien Nassau colliery, shafts about 12 ft. in diameter and 425 ft. deep were sunk in this way—namely, by boring with a rotary borer, a thick mud puddle being pumped into the shaft, so as to maintain a pressure in the shaft greater than the hydrostatic pressure in the surrounding loose sands and thus keeping the sides of the shaft from collapsing. He will find references to this method in *Das Abbohren von Schächten; Handbuch der Ingenieurwissenschaften*, p. 134, also in the *Transactions of the Institution of Mining Engineers*, vol. xiii (1896–97), p. 155. The process appears to have been employed even before 1895, though I cannot give the exact date when it was first used.

* Received Jan. 5, 1915.

A. BEEBY THOMPSON, London, England (communication to the Secretary*).—Mud drilling has attracted world-wide attention, and all engineers associated with petroleum have closely followed its phenomenal progress. As an exponent of drilling Mr. Knapp's writings are known and appreciated on this side of the Atlantic, where we welcome hints and suggestions by practical operators.

I have recently been professionally connected with drilling work of an exceptional kind which has so far defied all efforts to overcome. The human element is too great a factor in drilling to be neglected, but there is reason to believe that several of the drillers engaged were experienced, cautious and thoughtful men; nevertheless they affirm that the troubles are unique and insuperable.

The strata are composed almost exclusively of clay with occasional small lenticles of sand. The clays are believed to be steeply inclined, near the crest of a fold possibly inverted, and the whole mass is permeated with slip planes along which oil occurs. There is no evidence of stratification in the exposed beds, and even surface landslides are frequent along ridges. All the sand lenticles are saturated with oil, and a certain amount of gas is evolved when a depth of a few hundred feet has been reached.

The almost clean clay yields an excellent slimy, viscid mud from which sediment does not settle on standing, and must closely coincide with the requirements laid down by Mr. Knapp. Early efforts with a percussion rig were abandoned as a column of casing could not be kept free any distance, and before 600 ft. was reached the size was too small to go farther.

The rotary was started with 16-in. bit for insertion of 12½-in. drive pipe, and no serious difficulties commenced until a depth of about 500 ft. was reached. Although the thickest mud was employed in circulation, caving or heaving, as the drillers termed it, set in for an hour at a time, and lumps of clay about the size of a hand would be continually ejected, necessitating the employment of several men to clear away the material. Occasionally a block would occur and the pump pressure rose until the obstruction was ejected with force, throwing the mud lumps over the rotary table.

Casing was inserted, but nevertheless only a few feet could be drilled when heaving again commenced, and on one or two occasions the drill rods were twisted off. If the drill rods were withdrawn the hole filled up several hundred feet.

In a series of five wells the following means was taken to endeavor to effect progress: Casing was inserted at intervals and not more than 20 ft. of open hole allowed. This had to be discontinued, as the casing could not be moved after about 400 to 450 ft. had been inserted. Smaller

* Received Jan. 28, 1915.

diameters to 6-in. lining tubes were tried to induce more rapid circulation and less area of hole, with no satisfaction.

The casing was itself furnished with a cutting shoe and rotated cautiously while a flush was maintained, but lengths were twisted off by excessive lateral pressure, possibly due to cavings throwing the tube out of vertical.

Finally special heavy flush-jointed tubes were supplied for use either as drill rods or lining tubes. For the former use chilled cast-iron fish-tail bits were screwed to the 8-in. flush-jointed tubes, the object being to smash these up when the desired or maximum depth was attained. As a possible alternative it was intended to drill with 4- or 6-in. drill rods and keep the 8-in. flush-jointed tubes with a shoe near to the bit.

The former project failed, as it was difficult to restart circulation after the brief interval required for adding a new length of tubing, and eventually lateral pressure became so great that the column was twisted in two. The second method was unsuccessful on account of constant caving or heaving which intermittently ensued and prevented progress. Long intervals were allowed to elapse with the pump in operation to remove the cavings, but still progress was impossible.

My personal idea is that the excessive cavings originated from disturbed steeply inclined beds permeated with small slip planes where gas and oil had accumulated. Certainly gas did issue with the circulation mud, proving that the pressure had not been quite overcome by the mud, but the quantity was not such as to lead one to anticipate much difficulty from this cause.

An alternative proposal in view was to use larger pumps and work under a higher pressure, but this entailed an additional outlay not immediately available.

Another cause of the failure of the rotary came under the writer's notice when visiting Colombia last year. Here soft disturbed Tertiary clay shales predominated, and almost endless quantities of material were raised with the mud flush. Eventually mud was observed issuing from the fissures in the ground around the well and operations were ultimately abandoned. Heavy gas pressures were in this case encountered.

Any suggestions from members with experience in mud drilling to deal with problems of the kind mentioned would be a welcome addition to the literature of this comparatively new branch of the oil industry.

I. N. KNAPP, Ardmore, Pa. (communication to the Secretary*).—We are under great obligation to Mr. Thompson for the discussion he has presented and the detailed accounts of difficulties met in drilling wells in various localities.

I trust that this will encourage others who have experience in such

* Received Feb. 26, 1915.

matters to further discuss the art of drilling, or better, to present papers to the Institute on the solving of well-drilling problems and thus accumulate a record helpful to all interested.

Mr. Thompson does well to call attention first to the human element in drilling. This matter of vital importance is too often overlooked. A driller may be a very satisfactory man in a district he is familiar with, but this same man when put to drilling under new conditions may be an utter failure and become discouraged and homesick.

The fundamental requirement in all drilling operations to prospect for or produce water, brine, oil, or gas is to keep the walls of the well in shape so there can be no caving or distortion. Of course it may not always be possible to maintain this ideal condition.

When the walls of a drill hole are self-supporting no special problems are presented, but when such walls require support by using mud very serious problems arise.

The water used in drilling, if not sufficiently laden with mud, may cause erosion and thus start caving of the walls of the wells, or the water may slack the wall material, or the walls may be plastic, and under sufficient pressure will flow into the drill hole. The whole matter may be further complicated by the presence of oil or water associated with gas under pressure. Also nodules, concretions, or fossil wood may deflect the drill.

It needs united effort to solve these problems and I therefore most heartily indorse Mr. Thompson when he says, "Any suggestions from members with experience in mud drilling to deal with problems of the kind mentioned would be a welcome addition to the literature of this comparatively new branch of the oil industry."

The difficulties Mr. Thompson describes remind me of those I met and mistakes I made in the first few wells I drilled with the rotary using mud, although I employed drillers who were successful in other places using this method.

In the first place, I did not appreciate the necessity of keeping a mud circulation up to a standard that would not erode any of the soft sands encountered, and I now know the drillers did not.

Another mistake was that I allowed the free use of the mud circulation when not rotating and with the bit away from the bottom, the idea of the driller being to clear the hole of cavings and cuttings.

We found this was all wrong. When the drilling is skillfully done with a proper mud mixture the drill pipe will run in freely until the bit is on bottom. It should then be pulled back a foot or two until the pump is started.

No mud should be pumped into the well until the pump pit and surface mud are properly tempered. As soon as the well circulation is started the rotation of the drill should also be started. If the bit is

left for a long time at any one point in the well without rotating and with a strong mud circulation through it there may be a hole of considerable size washed out from the walls of the well and from which caving is liable to start. Slow rotation of the drill pipe when pumping with the bit off the bottom is necessary to keep the hole cylindrical and stop formation of irregular cavities.

Also, if a drill pipe hangs stationary in a drill hole it undoubtedly lies against the walls in places. Such lines of contact must deflect the circulation to the opposite side of the well and tend to enlarge the hole in that direction and so deform its shape; hence the necessity of some rotation with circulation.

Suppose that in running the drill pipe into a well, in which it was necessary to use mud, the bit meets an obstruction in the open hole 200 ft. from bottom and 300 ft. below a string of casing. If now the pump is started to flush the well out with mud, rotation should be started. If at the same time lumps of clay with sand should appear in the overflow I would consider the mud pump was running too fast and would slow down until the caved material stopped coming. I then would thicken the circulation and try to build up with the mud and stop the caving.

Naturally, the more you flush caved material out of a drill hole in soft ground the more extensive the cave becomes.

Another mistake made was not using a large enough drill coupling back of the rotary bit. At first we used a common 6-in. pipe coupling, $7\frac{1}{2}$ in. outside diameter by $5\frac{1}{8}$ in. long, into which to screw the 6-in. shank of a 15-in. bit. In such a case the bit could be deflected $3\frac{3}{4}$ in. from making a straight hole before the coupling would act as a guide. It is easy to guess that a hard nodule, a concretion, or a chunk of fossil wood, which are common in soft formations, would throw the bit and drill pipe much farther out of line than indicated above and leave a decided hump in the drill hole.

After pulling out and running the bit in again it may hit the hump and cut or knock off the deflecting material and plug the hole. This I believe did happen in my drilling several times, for on rotating to drill out the obstruction probably a new and straight hole would start. We have lost as much as 300 ft. of hole at one time from this probable cause.

A hole drilled with too small a coupling back of the bit is usually more or less humped and crooked and cannot be cased without running in a toothed rotary shoe or a reamer to straighten the hole.

In a 15-in. hole the outside diameter of the drill coupling should be 12 in. by 18 in. in length to be of sufficient size to form a guide to the bit to keep the hole straight and cylindrical and prevent humps.

We made many other mistakes, but they were not as vital to the completion of the hole as those described.

If a hole is not allowed to cave in the least and is kept straight and cylindrical, casing and cementing troubles disappear.

We had twist-offs, collapsed casing, and other troubles, but as we became more skillful, these also practically disappeared.

I cannot concur with Mr. Thompson when he says "My personal idea is that excessive caving originated from disturbed steeply inclined beds permeated with small slip planes where gas and oil had accumulated."

I believe that in any sized hole up to 24 in. in diameter the caving could only have originated from the mistakes in drilling I have outlined. When the cave is once started and gets to be of considerable dimensions, say 4 ft. or more, then Mr. Thompson's explanations would apply.

To illustrate the difficulties that have been overcome in the past, I will quote from I. C. White in the *West Virginia Geological Survey*, vol. i, p. 147 (1899), as follows:

"Soon after the Burning Springs (W. Va.) oil development began in 1860 the petroleum craze spread all over the State. . . . Many wells were drilled in several counties or at least attempts were made to drill them, which nearly always ended by getting the tools fast, and the hole plugged, because the operators had not yet learned the art of dealing successfully with rocks that crumble, or cave, and fall into the hole when water touches them. In the region of Titusville, Oil City and all of north-western Pennsylvania the rocks (sub-Carboniferous and Catskill) to be drilled through are all hard and the walls of the wells stand firm after the holes are bored, even though drilled 'wet' and full of water from top to bottom. But when the Pennsylvania drillers came down into West Virginia where a much higher and softer series of rocks was encountered (Permian and Coal Measures) and attempted to use the Pennsylvania methods, the result in most cases was failure to sink the borings to any of the Venango County oil-producing sands. Thus it happened that the oil development of West Virginia outside of the Burning Springs and Volcano 'oil breaks' or anticlinal was delayed for 30 years behind that of her sister State on the north."

Cable-tool drilling in both Pennsylvania and West Virginia is carried on by rigs, machinery, and tools of practically the same design for the same size and depth of hole. It is a matter of experience for the driller to use the proper method and manipulation in handling the different ground to get the desired results.

The hydraulic rotary method of drilling with the use of mud when necessary is capable of doing a very much greater range of work than is possible with the cable tools. It is again a question of experience of the driller.

The average driller is very conservative and sticks to what he knows how to do and instinctively tries to use methods he is familiar with to fit new conditions when they may not be at all applicable to the case in hand. He is very resourceful so far as his experience goes.

The oil industry has spread out to such a degree that it has greater need than ever for drillers capable of solving the problems of new fields and their new and variable conditions.

The art of drilling I think offers a great field with ample financial reward to the young engineering graduate who is willing to go into the derrick and work and acquire the "feeling" necessary to really judge of the ground drilled and check the work of the driller.

In many arts it is not at all necessary for the engineer to become an artisan in order to be entirely competent to manage every detail, but drilling does require it.

The Rôle and Fate of the Connate Water in Oil and Gas Sands

Discussion of the paper of ROSWELL H. JOHNSON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 221 to 226.

A. C. LANE, Tufts College, Mass.—About a year ago Mr. Washburne read a paper on the same characteristic oil-sand waters, discussing their chemical character, and the possibility of their chemical character being modified by adsorption in passing through the shale. We are used to consider circulation as being in one stratum, and I have often gone so far as to think that 2,000 or 3,000 ft. down I could recognize the chemical character as in part connate to a given water. For instance, the St. Peter sandstone, far underneath oil sands, seems to retain a relatively fresh and peculiar character. I am, therefore, tempted to ask the writer if in his experiments he has made any test to see whether the chemical character of these waters is also changed, where there is any adsorption, so that the water passing through these fine-grained beds should leave behind its salt as well as its oil. In many cases salt waters are associated with oil and gas.

A second question has occurred to me. In these days when we hear a good deal about oil flotation, may it not be well to repeat these experiments with pyritic concentrates, and see if they observe precisely the same with regard to the selective accumulation of oil and gas.

ROSWELL H. JOHNSON.—I thank Professor Lane for bringing this question forward. I recognize its importance, but am not able to say I have any data upon that point. It is also very interesting in connection with Dr. Day's hypothesis of fractional filtration, and we may very easily get something happening to the water corresponding in some degree to what we get with oil.

D. B. REGER, Morgantown, W. Va.—Mr. Johnson in his paper states his belief that the oil is formed originally from organic matter in the shale, and is forced out of the shale into the porous reservoir of the sand above the shale. He states further, that as you go deeper in the measures where there is an increasing overburden, the oil becomes less and the gas increases in large proportion, or he says that the oil is forced back out of the sand by pressure, into the shale.

Now the oil evidently, in the first place, is forced out of the shale, because there is a lack of pores in the shale, and is forced into the porous reservoir of the sand.

I ask Mr. Johnson why he believes that the oil in the lower measures, where there is more overburden, is forced out of the porous reservoir, or sand, back into the shale, which we know ordinarily does not contain pores of any kind. I cannot see why it would be forced out of the top of the sand into the shale above, when there is only a little overburden; then, reversely, forced back out of the porous reservoir into the shale again after you have a greater weight of measures deposited above it. His statements are to the effect that oil can be forced out of a porous sand into an impervious shale.

MR. JOHNSON.—It is unfortunate that in this paper I have been discussing two ideas, and I believe this confusion arises from that fact. I have been talking, on the one hand, of the compacting of the shale, driving its contents along, and then this segregating work which takes place as it passes large pores. Another very different thing, is that after the pressure becomes fairly great, a new dynamo-chemical gas begins to form, and this gas, moving into the reservoir, displaces the oil. We are dealing, in the first place, with a purely mechanical feature, a mere pressing out of the fluid contents of the shale, no new gas being formed at all; then later we have this new gas being formed dynamo-chemically, which, going in there, and being held by selective segregation, has to displace the other contents.

I. N. KNAPP, Ardmore, Pa.—In regard to the formation of the gas, I have no theory about that, but it is a practical matter that in Louisiana, between 1,600 and 1,700 ft., I got a clear dry gas under a pressure of 650 lb. to the square inch. I went down below that, as deep as to 3,000 ft., and I know of wells being driven to 4,500 ft., in these formations. The fossils saved from wells, up to 2,400 ft. were submitted to Prof. G. D. Harris and Dr. Maury. These fossils indicated to them that we had not reached the Tertiary, that is, that the gas was very recent. An analysis showed that the gas contained 98 to 99 per cent. marsh gas. The drill shows many porous reservoirs that hold oil and gas in paying quantity that are absolutely dry. As for oil and gas traveling through the ground, I have come across conditions where they could not travel through the ground as they were imprisoned in globules in salt and magnetic iron, where there was absolutely no porosity.

Communication to the Secretary*.—After making some laboratory experiments and referring in a rather general way to various theories advanced by others, Professor Johnson offers four tentative hypotheses as having a bearing on the practice of oil prospecting.

* Received Mar. 3, 1915.

I will attempt to discuss only one of these "practical conclusions," as follows: "First, gas is not to be expected in large quantities and under high pressures in relatively recent unconsolidated deposits."

As the title of the paper relates to a purely theoretical matter and the conclusion quoted relates to practical prospecting only, I am sure I will be pardoned for not discussing the theory.

In looking for oil indications along the Gulf Coast I found there were very many gas escapements in Terrebonne Parish, La. A few were found highly impregnated with sulphuretted hydrogen. I had numerous samples taken for analysis and the result seemed to indicate that oil might be found by drilling.

Fig. 2 shows a typical gas escapement in a field.



FIG. 2.—GAS ESCAPEMENT IN A FIELD, TERREBONNE PARISH, LA.

These escapements are fairly constant in volume and have been known for many years.

The discovery of deep gas in the Parish mentioned was made on the Lirette farm in Sec. 51, T. 19-S, R. 19-E. (see *Bulletin* 429, *U. S. Geological Survey*, p. 163).

This well blew out from 1,710 ft. and continued to blow for 99 days at an estimated rate of 5,000,000 cu. ft. per 24 hr. without any apparent weakening, and then it caved in and shut off. The first 24 hr. it made dry clean gas and then began to spray salt water.

Previous to the blowout of the Lirette well and about 5 miles from it, I drilled a prospect well over 2,400 ft. deep in unconsolidated material. I submitted fossil shells from this and other Terrebonne Parish wells to Prof. G. D. Harris and Doctor Maury (see *Bulletin No. 429, U. S. Geological Survey*, pp. 169 to 173). They definitely state "that none of these wells have penetrated beds that can be regarded as older than Pleistocene." I presume therefore it will not be questioned but that these are geological speaking "relatively recent unconsolidated deposits."

After the Lirette well blowout I drilled four wells within about a mile of it in an effort to make a commercial development. Gas in quantity and under great pressure was found in each well from the same probable horizon.

Below 1,500 ft. thin layers of slightly cemented sands were found that were probably true gas sands and these were bedded with thin layers of loose sand, black clays or shales, and shell conglomerates. There were salt-water sands above and below near the gas, but I was unable to locate them definitely. Below 700 ft. the clays were very dry and these dry clays below 1,200 ft. would flow when the pressure was taken off by bailing a well and gradually close it up.

Two of the four wells mentioned were lost by blowouts that lasted about 24 hr. before the wells caved and shut off. I feel confident that each of these wells was making gas at a rate exceeding 20,000,000 cu. ft. per 24 hr. for some hours before it shut off. I base this statement of quantity on observations made previously on many wells of measured capacity.

I saved one well as a gas well and it gave dry clean gas for a couple of months, when it was shut off by clay squeezing in. This was in the 1,600 to 1,650 ft. horizon. The closed pressure was 650 lb. and the initial open flow capacity 2,000,000 cu. ft. per 24 hr. An analysis showed 98 per cent. marsh gas and no illuminates, which I think indicates the entire absence of oil. There was not the slightest taste or odor of petroleum.

I believe I have shown by the drill that gas may exist "in large quantities and under high pressure in relatively recent unconsolidated deposits" and this is diametrically opposite to the author's first conclusion.

There have been numerous gas blowouts from recent deposits in the Louisiana and Texas oil fields contiguous to and over productive oil pools. I witnessed several such blowouts from comparatively shallow depths and their violence indicated the gas was under high pressure.

I can refer to many published accounts supporting my showing. For instance, in *Petroleum Mining*, A. Beeby Thompson, under "Evolution of Gas" (pp. 94 to 100) cites many instances of large volumes of gas

escaping from unconsolidated material in the various oil fields of Russia, Burma, Borneo, Peru, and Trinidad.

One view (Plate XIII) shows where "Many thousand tons of puddled clay were ejected in a few hours, accompanied by the evolution of hundreds of thousands of cubic feet of inflammable gas."

The *New Orleans Times-Democrat* of Nov. 26, 1911, gives an account of a gas eruption in the sea off Erin Point, Trinidad, on Nov. 11, 1911, and says "The next day it was found that an island of $2\frac{1}{2}$ acres had been formed. A landing party found the place still warm, and by laying down boards were able to examine two cones, 12 to 15 ft. high, from which gas was escaping. The air was saturated with the odor of sulphur and oil."

An observer of this eruption told me the gas made a flame over a mile high for perhaps half an hour. His estimate of height he checked by a rough triangulation.

Fig. 3 shows typical mud cones of medium dimensions such as form in the oil and pitch lake regions of Trinidad and Venezuela.



FIG. 3.—TYPICAL MUD CONES FORMED BY GAS, TRINIDAD, BRITISH WEST INDIES.

Natural gas is continually forcing up cones of pitch in the pitch lakes and away from them cones of clay or mud, and they break at the apex and gas issues that may be lighted and sometimes burns for days. Occasionally pitch and oil show with the gas. I have this information from a man who lived for some months in this region and drilled some deep wells in Trinidad. He also said the gas found below the surface increased with the depth and pressures of between 400 and 500 lb. were expected around 1,200 ft. in depth.

Peckham reports on the pitch lake of Trinidad as follows: "As the bitumen rises in the center of the so-called lake it is inflated with gas . . . the gas cavities are of all sizes, some of them very large, and in the aggregate occupy, at a rough estimate, from one-third to one-half the volume of the pitch." (See *American Journal of Science*, 4th ser., vol. i, p. 193.)

Mud lumps are common in the delta of the Mississippi. The mud slowly accumulates around the gas and sludge vents and forms cones ranging in height from a few inches to several feet. These lumps are largely composed of structureless clay. (See *Professional Paper No. 85-B, U. S. Geological Survey.*)

It is very likely the gas would form cones in Terrebonne Parish under proper conditions, but the drill shows numerous loose sand strata and shell beds capable of relieving in a lateral direction any tendency of a rise in gas pressure sufficient to form cones.

I am sure my references also show that gas may be expected in large quantities in relatively recent unconsolidated deposits.

A Modern Rotary Drill

Discussion of the paper of HOWARD R. HUGHES, presented at the New York meeting, February, 1915, and printed in *Bulletin No. 99*, March, 1915, pp. 629 to 635.

I. N. KNAPP, Ardmore, Pa.—The claims made in the paper for the Sharp & Hughes cone bit are modest. I have used it with satisfactory results. It is a wonder for eating through hard, brittle rock; it will drill through iron pyrites that the fish-tail bit cannot enter.

Soft ground is apt to contain concretions that are much harder than the containing ground. We have at times in drilling with a fish-tail bit thought the hole was partly in such hard material and partly in soft, thus making a difficult place to drill past in the ordinary way. By changing to a Sharp & Hughes cone bit such obstructions were easily overcome. This cone bit has greatly widened the range of work that may now be profitably accomplished by the hydraulic rotary method of drilling.

It is not, however, adapted to drill soft or sticky ground, the old fish-tail bit being better for such purposes. The fish-tail bit usually scrapes its way through the material drilled, but not always. I believe at times it cuts and chips in soft friable rock. That is, the bit seems to bite into the rock, and when sufficient torsion is put on the drill stem it breaks its hold and jumps around and chips the rock from the bottom of the hole, like a stone cutter's chisel, and then takes another hold. This condition occurs only in friable material when the bit is sharp and the feed properly controlled. The Sharp & Hughes cone bit is a most valuable tool for the rotary driller.

Improved Methods of Deep Drilling in the Coalinga Oil Field, California

Discussion of the paper of M. E. LOMBARDI, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 209 to 215.

I. N. KNAPP, Ardmore, Pa.—Some years ago I made a swinging spider to carry a string of casing in a way somewhat similar to that described in this paper. The ground I tried it in was too soft for its successful operation.

The spider had a central one-flange bushing to carry the slips. Under this flange and on top of the spider was a properly protected ball runway, or ball bearing, which permitted rotating the casing when it was in place and supported by the slips.

It is advantageous when a casing is moved up and down to keep it free to be able to turn it slightly, say one-quarter to one-third of a revolution, at the top of the stroke. This rotation tends to keep a better clearance for the casing than a simple up-and-down movement.

Gasoline from "Synthetic" Crude Oil

Discussion of the paper of WALTER O. SNELLING, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 100, April, 1915, pp. 695 to 704.

A. F. LUCAS, Washington, D. C.—Are the results of the experiments in practical operation?

WALTER O. SNELLING.—There have been no large-scale operations up to the present. The necessary conditions of temperature are easily arranged for, but the required condition of pressure, 800 lb., is not so simple a matter. The company has been working on this problem, and at the refinery in Oklahoma work of this kind is now being carried on. The way in which an ordinary still will leak at 50 lb., and more at 100 lb., leaves some doubt in the mind of many as to what can be done at a continuous working pressure of 800 lb. On the other hand, the fact that they are making ammonia abroad by a method which employs higher pressures and temperatures than anything involved in this method, leads me to believe that the difficulties, which will undoubtedly arise when this is attempted on a commercial scale, can be overcome.

WILLIAM N. BEST, New York, N. Y.—Is there any change in the calorific value of these oils?

Mr. SNELLING.—I can only answer that in this way—in one of our experiments we took 300 cc. of paraffin, and the product which came from it amounted to 305 cc., an actual increase in volume, which is what one, as a rule, expects, when a material that is heavy is changed into a sub-

stance of lighter gravity. The number of heat units present in the original 300 cc. of paraffin were all present in the final 305 cc. of synthetic crude oil, and the heat units were in that proportion in the final product. The gasoline produced varies in gravity. This is determined by where we cut it off. We always aim at 70° B_é. The tests of the gasoline with engines, and in burning, indicate that it is apparently quite similar to ordinary gasoline, but wholly different from the cracked product produced by passing oil vapors through tubes at ordinary pressures. It is similar to, if not absolutely identical with, the hydrocarbons normal in the gasoline from ordinary crude oil.

F. G. COTTRELL, San Francisco, Cal.—What temperature do you use?

MR. SNELLING.—The criterion we generally use is pressure, for the reason that in laboratory experiments the temperature is more difficult to determine in the interior of a vessel of this kind, but of course the temperature corresponds to the vapor pressure produced.

M. R. WOLFARD, Cambridge, Mass.—Was any method used to exhaust the air, and has the air above the oil any particular influence on the process? I think it would be interesting if we had definite determinations of the heating value of the oil. It is rather peculiar you can get from 100 cc. of oil 105 cc. of crude oil which normally has a higher heating value than heavier oil. It would seem necessary, therefore, to make calorimetric tests in order to demonstrate the fact positively that there is no change in the heating value. It would also be interesting to know whether synthetic gasoline always has precisely the same heating value although the density of the original oil may be different.

MR. SNELLING.—In regard to the first matter, I may say that one of my assistants has used natural gas and other gases in the remaining part of the vessel not filled with oil, and we have also made tests with that space evacuated, but we could not see that it made any great difference, except that the exclusion of the air led to better results, for the reason that, as our criterion was pressure, naturally if there was no original pressure in our vessel, the temperature used was somewhat higher.

In continuous processes, the gases occupying the space over the oil play a very significant part, and their composition is a matter of considerable importance. Fortunately, however, this is a matter which adjusts itself very nicely, and since the gases over the oil are a portion of the equilibrium system existing, any change in the conditions of equilibrium leads to a corresponding change in the composition of the gases present.

As to the second point, it is a pretty well recognized law of thermodynamics and conservation of energy, that with a given mass of material

placed in a vessel, and nothing going out, and with all the material which was placed in the vessel still in it when you get through, there will be no change in total energy. If we start with 300 cc. of paraffin, when we get through, we have 305 cc. of the synthetic crude oil. The gravity of this synthetic crude oil is lighter than the paraffin which went in. The number of heat units in the synthetic crude oil can scarcely be considered to be different from the number of heat units which went in, because according to the conservation of matter, you have a certain number of carbon atoms and hydrogen atoms, and it is the burning of this carbon and hydrogen which gives the heat.

DAVID T. DAY, Washington, D. C.—This matter is very interesting. It is evidently in a very unfinished state yet, as Mr. Snelling admits, but I think we should be very grateful to him for bringing this preliminary evidence before us so promptly and so helpfully. While there is a great deal of work going on on this subject in many different lines, there are many interesting results, and these results must be classed among them.

There are several small matters in regard to which I would like more information. I am deeply interested in the fact that certain products were obtained here under certain conditions; for example, that if you had the bomb almost full of yellow lubricating oil, say three-quarters full, and then heated it so as to produce a pressure of 800 lb. to the square inch, you would not change it from that yellow oil. This is the most remarkable statement that has been made. We have this fact, that the proportion of liquid, to begin with, in the bottle must be three-elevenths to get the maximum effect, and if you have it three-quarters full you get no effect whatever. That is extremely remarkable, and I would doubt whether continued experiment would always bring that same result, because the temperature of 300° or 250° and 800 lb. pressure of that oil in any kind of vessel is going to change the oil, I think, and perhaps that result was more or less accidental, particularly in view of the fact that you got no change whatever under these other conditions. As long as you can change oils when you heat them and compress them in this way in tubes and in bottles, etc., in so many different ways, and the fact that the odors of these oils are not those of crude oils, I fail to see the importance of the generalization that Mr. Snelling is inclined to make that this has some relation to the crude oils in the earth. I doubt whether the natural light-colored oils have been changed in nature into dark-colored oils by such a process as this, as would be inferred from the generalization made by Mr. Snelling. I fail to see the force of that. I do not think that Calgary gasoline has been changed into heavier oils where we find them in Canada.

If after heating an oil under pressure this way, and obtaining so much light oils, the light oils are distilled off, and the residue then treated the

same way, and that process is repeated several times until, say, 70 per cent. of the oil had been converted into gasoline, what percentage of coke was obtained?

MR. SNELLING.—Dr. Day's question brings up several interesting points I did not take time to speak of, because I feared to trespass upon your patience. In the first place, in regard to the contention that it is not possible under these conditions to heat an oil and have it remain but little changed: If we take, for example, some gasoline, or any light oil, you can put it through this process and it will come out wholly unchanged, and you will be able to see the reason why, since the criterion is not temperature at all, but pressure. A pressure of 800 lb. can be produced by the heating, and the gasoline be wholly unaltered chemically. In my earlier workings with liquefied natural gas, we used a nominal pressure of 200, 300, or even 500 lb., in bottles of this gas. I have taken these bottles and placed them in warm water in order to test the containers, under the Interstate Commerce Commission specifications, and even after a pressure of 2,000 or 3,000 lb. per square inch was reached, the gasoline was wholly unchanged. The low-boiling material, heated to a point at which it was quite volatile, produced by its volatility the high pressure registered. No chemical change occurred in such case, as can be readily seen.

MR. DAY.—What temperature, 300° C.?

MR. SNELLING.—The temperature in our experiments with the conversion of hydrocarbons is such as produces within the vessel a pressure of about 800 lb. per square inch.

MR. DAY.—Do I understand that a yellow lubricating oil, an oil like that, has ever been heated in a bomb to 800 lb. pressure and to a temperature of 250° to 300° C., and remained like that? The temperature feature being essential to the discussion I have made.

MR. SNELLING.—As I have not a complete set of my experiments here, I can answer that in this way—that we have, for the purposes of our own records, made many experiments, using pyrometers in the tubes of these vessels. We have the correlation reasonably exact between temperature and pressure. There is always, when oil is heated, more or less of a change, and that change will result in causing a production of tarry products, etc. That yellow oil, when it occupies three-quarters of the volume of the treating vessel, may be heated to a high temperature, without producing material which resembles crude oil or contains any perceptible amount of uncracked gasoline. If that oil be placed in a vessel and that vessel be three-quarters full, and be brought up to the temperature necessary to bring about cracking, then of course it will be altered, and it will undergo changes which will cause the color to change, but there will be no considerable production of gasoline.

LEONARD WALDO, New York, N. Y.—Does the temperature enter into this reaction?

MR. SNELLING.—In many of our experiments, the habit of my assistant has been to bring the vessel up and watch the gauge, and when it reaches 800 or 900, simply to take it off, and push the vessel under a stream of cold water. We made some experiments in which we continued the heat. We found a little different result. It is, apparently, a reversible reaction. As soon as the conditions are favorable for this change to go on, it goes on, until it can no longer go on; in other words, it goes on to equilibrium. This change apparently goes on until a sufficient amount of material, crude oil, is produced, so that each one of its components has a certain partial pressure. You can see how that must be—as soon as a certain amount of gasoline is produced, that temperature alone causes this gasoline to have a certain back pressure. If the vessel were then to remain over night at that temperature, I feel sure, from the results of our experiments, there would not be much further change, and when the material came out it would be quite the same. In chemistry in the last few years we have become familiar with catalytic action and have learned that certain reactions which take place do so under certain given conditions. We also learn that any reaction which will take place at all can have its reaction velocity facilitated or quickened by the presence of certain catalysis, and almost any reaction which will take place with the catalytic mass in quick time will take place in itself if a sufficient length of time is given; in other words, the same set of conditions that produce a material in an hour or in a minute in the laboratory, could, at a far lower temperature, provided the same fundamental condition or underlying condition existed, produce the same effect in a longer period of time. We are familiar with sets of experiments in which a certain thing goes on to equilibrium with catalysis in 5 min. at a high temperature, and that same thing would take weeks, months, or years to occur at low temperatures, and therefore one significance of this, as shown by these synthetic petroleums, is their similarity to natural crude.

MR. LUCAS.—Can you connect any relationship in your experiments with the origin of oil?

MR. SNELLING.—Only to this extent, that if things so diverse in characteristics as kerosene, rod wax, fuel oil, or paraffin, when put under this given set of conditions give always a product which resembles one thing, natural crude oil, then it appears to me not improbable that this similarity may have significance, in view of the fact that we have reasons to believe that petroleum may have been formed in the earth from many different materials. Let me take a somewhat analogous case, in which our society has played such a remarkably prominent part. Not many years ago, the origin of ore deposits was a subject but little understood. One set

of men, who were familiar with the ore deposits in certain mining districts, insisted that ore deposits had been formed by uprising mineralized waters, coming from the depths of the earth. Other men contended that ore deposits were formed by the leaching and segregation of the minute amounts of heavy metals present in the adjacent rocks. The two magnificent volumes which this society has published on ore deposits and which form unquestionably the most important contributions which have ever been made to our knowledge of the subject, show us rather conclusively that both these sets of men were wholly right. Some ore deposits have been formed by mineralized waters, from great depths, and some mineral deposits have been formed by the leaching and re-deposition of minerals, from the very slightly mineralized rock near the surface of the earth.

To-day there are many prominent oil men who believe that petroleum has come from certain transformations of vegetable or animal matter within the sedimentary rocks of the earth's surface. Others are contending that petroleum has a deep-seated source, and that this source is quite likely related in some manner to the barysphere of the earth. It is my belief that the coming years will prove, just as has happened already in the case of ore deposits, that all of these gentlemen are right. The petroleum in certain known deposits has unquestionably been formed by certain changes which have come about in accumulations of vegetable matter. Other petroleum deposits unquestionably owe their origin to the animal remains present within certain shales with which these oils are associated. Still other deposits of petroleum, I believe, have had a deep-seated source, and have originated through the chemical transformation of gases and vapors coming from comparatively great depths. Since the studies which I have outlined in this paper show that materials of such different appearance as fuel oil, kerosene, and paraffin may all under perfectly definite conditions of temperature and pressure change into a material resembling crude oil so closely that even an expert may be deceived, then at once we can see why all the generalizations which have been drawn in regard to the color, gasoline content, bloom, etc., of oils, as proving something about their origin, are wholly illusory. Starting from the position that practically all hydrocarbons may, under certain well-defined conditions, produce a material which we may consider to be crude oil, it seems to me that we stand in a favorable position to review our existing knowledge in regard to petroleum deposits, and to weigh in a fair and unprejudiced manner the arguments which all the parties have brought forward, favoring either a vegetable, a mineral, or an animal origin for petroleum. Personally I believe that petroleum can be, and has been in earth history, produced from all three kingdoms, mineral, animal, and vegetable.

ROSWELL H. JOHNSON, Pittsburgh, Pa.—From a geological standpoint, David White's recent work, which was given before the Geological Society of America in December, seems particularly interesting in the light of this discussion. He found that the quality of the oil varied with the type of organic compounds in the accompanying shales—that in Pennsylvania as we moved toward the district where we got our harder coals, those with higher fixed carbon proportions, that there also the organic matter in shales was higher in fixed carbon, and that the petroleum varied step by step also. I do not venture to say what the interrelation of these two things is, at this time but all these hydrocarbon transformation experiments must now take on a new interest to the geologist.

The Estimation of Oil Reserves

Discussion of the paper of CHESTER W. WASHBURNE, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 469 to 471.

ROSWELL H. JOHNSON, Pittsburgh, Pa.—I hope that this contribution of Mr. Washburne's is only the first of a series dealing more in detail with the very important problems involved.

Pores are of two kinds: Those which are entirely inclosed, so that they do not belong to a system of communicating pores tapped by the bore hole or shot hole; second, those which do so communicate. Porosity, of course, includes both. But since pores of the first class are valueless to us, we should confine our attention to the latter, for which I propose the term "effective porosity." This should be measured by the Wisconsin method, based on the quantity of fluid that can be introduced by the aid of vacuum.

Another factor which requires attention in this connection is the dependence of the extractability upon the rate of loss of pressure, as affected by the number of holes, and the rapidity of their completion, the existence of uncontrolled wells, the use by other operators of vacuum pumps upon the casing head, the exhaustion of a gas field in the same reservoir prior to the discovery of the oil, etc.

The extractability depends upon the pressure available for the exclusion of the oil from the reservoir into the hole. The action of gravity in bringing oil into the hole is relatively small where the pores are very small, as is so frequently the case.

A Study of the Chloridizing Roast and Its Application to the Separation of Copper from Nickel

Discussion of the paper of BOYD DUDLEY, JR., presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 96, December, 1914, pp. 2767 to 2782.

H. O. HOFMAN, Boston, Mass.—At the close of the paper Mr. Dudley said that his experiments in the laboratory had shown that it was possible to separate nickel from copper by means of a chloridizing roasting, but that in order to demonstrate a process of this kind, work upon a large scale would have to be carried out.

I think it was in 1913, that experiments of this kind were carried out at the works of the Pennsylvania Salt Manufacturing Co., Philadelphia, in a Wedge furnace, and at Sault Ste. Marie in a Sjöstedt furnace.

They showed a satisfactory separation of nickel and copper. The ore was first subjected to an oxidizing roast, then mixed with pyrite and salt, and given a chloridizing roast.

A. H. Carpenter published in the *Engineering and Mining Journal*, vol. xcvi, p. 1085 (1914) the details of this work. The conclusion arrived at is, that if you subject sulphide ore to an oxidizing roast, finishing at the temperature at which all nickel sulphate is dissociated, i.e., 764°C ., then mix it with pyrite and salt, and subject the mixture to a chloridizing roast between 600° and 700°C ., you can chloridize a high percentage of the copper without chloridizing any of the nickel, and obtain thereby a satisfactory means of separating the two.

These experiments have also shown that if in the oxidizing roast the temperature is not carried sufficiently high, so that water-soluble nickel sulphate remains in the ore, this nickel sulphate will be chloridized.

Effect of Zn_3Ag_2 upon the Desilverization of Lead

Discussion of the paper of F. C. NEWTON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 473 to 477.

H. O. HOFMAN, Boston, Mass.—The results of these large-scale experiments, which were carried on at Perth Amboy, show that there are many differences between theoretical conceptions and their applications to work on a large scale.

The author has given reasons why the results of bringing a desilverizing kettle to the high melting point of the zinc-silver alloy, Zn_3Ag_2 , melting at 665°C ., are not satisfactory. In all probability the alloy is dissolved in the lead; it may also be decomposed by the lead; the practical result is that this high temperature does not work. It remains now for the laboratory man to carry on systematic experiments

to find out what takes place when this zinc-silver compound is brought into contact with lead at various temperatures.

I have in mind taking a number of iron tubes, mixing the alloy, Ag_2Zn_3 , with lead, bringing the mixtures to different temperatures, holding them there for a given time, then chilling the tubes, opening them, and making assays and micrographs of cross-sections of the lead cylinders.

J. W. RICHARDS, So. Bethlehem, Pa.—The curve on p. 476 would show the results more clearly if the temperatures were taken as abscissæ and the grade of the bullion was carried as ordinates. In that way the curve would not have so irregular a shape, and the results would be more easily seen.

W. MCA. JOHNSON, New York, N. Y.—I would like to know a little more about this compound of zinc and silver.

F. C. NEWTON.—This compound was indicated by Professor Carpenter's work upon this zinc-silver lot of alloys, but I have never seen or made the alloy. I accepted his work and went ahead on that basis. I know very little of the properties of the compound except that it contains about 53 per cent. of silver. As a matter of fact, I have found some very interesting alloys in the retort metal, that is, the residue left in the retort after the zinc. In the retort metal, the average contents being 3,000 oz. of silver, I have taken lumps as big as or larger than a walnut of an alloy that would assay 11,000 oz. In retorting, a crust will form that runs a little high in copper if the operation is not done at the highest heat, that is, if the draft conditions are not exactly right; a hard zincky skin will form on that retort metal, which will run anywhere from 8,000 to 9,000 oz., whereas the ordinary grade of retort metal would be from 4,000 to 5,000 oz. So it is clearly indicated that these higher compounds of zinc are formed and form in the lead too. The compound is considered highly undesirable as you cannot get the zinc out of it, and the zinc losses, which ordinarily are 3 to 4 per cent. of the zinc in the crust, under these conditions would go up to 10 per cent.

MR. HOFMAN.—I have not had any actual experience with this chemical compound. All I know of it is that Professor Carpenter drew the freezing-point curve of the alloys of zinc-silver and proved the existence of the compound. He did not take up the description of the compound nor its behavior when subjected to different reagents and different temperatures. I do not know that anything is known definitely about that compound.

MR. JOHNSON.—Is it soluble in lead?

MR. HOFMAN.—Professor Carpenter did not touch on that question. That must be investigated. As I said a moment ago, in a year from now I hope to have some data on this question.

MR. JOHNSON.—How about the compound being analogous to yellow brass, two parts copper and one zinc?

MR. HOFMAN.—The freezing-point curve of the zinc-silver alloy resembles very much that of the copper-zinc alloy. It is a curve which passes from the freezing point of silver downward to the freezing point of zinc; there is but one decided break, which shows that there exists a chemical compound. As to the behavior of the zinc-silver compound and its resemblance to that of the copper-zinc compound, I have no information.

MR. JOHNSON.—Is yellow brass supposed to be a definite chemical compound or an indefinite one?

MR. HOFMAN.—As I recall the copper-zinc curve there exists a single intermetallic compound, Cu_2Zn_3 , with copper 39.33 and zinc 60.67 per cent.; standard yellow brass contains copper 66.6 and zinc 33.4 per cent. The chemical compound Cu_3Zn_2 corresponds to Muntz metal, copper 60 and zinc 40 per cent.

MR. JOHNSON.—As a profitable research I would suggest that the boiling point as regards zinc be taken as a means of diagnosing the chemical state of these alleged zinc-silver alloys. A curve expressing such a set of determinations would give a sharp point when a change of molecular state occurred. Some years ago, I made some determinations of boiling point as regards zinc of brass, the two-thirds copper, one-third zinc alloy, commonly called yellow brass in the trade. This evolved zinc vapor at 760 mm. pressure, at $1,065^\circ \text{C.}$ as against 920°C. (the boiling point of zinc). Silver resembles copper so much that it is natural to conclude that silver-zinc alloys behave analogously to copper-zinc alloys. Such a research would throw light and give clear evidence of a practical and commercial nature on such hypothetical compounds as are believed to exist. It would also have a bearing on the distillation of "zinc crusts" in the Parkes process.

J. W. RICHARDS.—The presence of the supposed maximum on the freezing-point curve is an indication of the compound. Now, that compound can be made by melting these two materials together in these proportions, and then its solubility in lead can be studied.

In studying this question, however, as to the solubility of metals in each other and their compounds, it should be borne very strongly in mind that in the melted metals those compounds do not exist. You will have the best view of the melted metal by considering it as a solution of what is present there in the definite proportions in which they exist.

There is no evidence that in the melted metal itself there is any re-solution into compounds. The melt consists only of the melted metals

in mutual solution in the proportions in which they are present. The compounds are in solidified alloys, but it would be a mistake to imagine their presence in the melt. The question then comes up: What is the tendency to segregate from the melt during the operation of freezing?

EDWARD P. MATHEWSON, Anaconda, Mont.—There is one point brought out by Mr. Newton which was not brought out very strongly, and I think it has some bearing on the recent discussion. He stated there were some very rich silver alloys or compounds which segregated from the retort bullion, and he had taken out pieces as large as a walnut, extremely rich in silver, from bullion not very rich. And he also stated that these were also found when the bullion was not very free from copper.

I think that is the secret of the segregation of these rich alloys of silver; that the copper is there always. My recollection is that these rich alloys of silver were never found unless copper was present, and the copper had something to do with this alloy.

MR. NEWTON.—What Mr. Mathewson says is perfectly true. In refining lead bullion great care is taken to eliminate all the copper, as after the bullion is softened it is tapped into the desilverizing kettle, and there it may contain from 2 to 3 per cent. of copper. That bullion is brought down to the freezing point and skimmed, and in case of high copper it may be brought up and heated again and then re-chilled to get the bullion under 0.1 per cent. of copper.

The consumption of zinc is very high in case copper is present. The copper takes a certain portion of the zinc, and that zinc is not available for the extraction of silver; also, the difficulties of retorting are very apparent the instant the copper rises.

Copper Smelting in Japan

Discussion of the paper of MANUEL EISSLER, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 95, November, 1914, pp. 2661 to 2703.

J. W. RICHARDS, So. Bethlehem, Pa.—I do not see in the paper a description of the Mabuki hearth as used for ore smelting. I noticed it described for concentrating matte, but I believe the older Japanese process was to use it for smelting directly, by a pyritic smelting and Bessemerizing all in one operation, in the same simple hearth apparatus. There is a monograph published in German, on Copper-Ore Smelting in the Mabuki Hearth.

At Ashio the practice of injecting bituminous coal at the tuyères is noteworthy. It is very interesting to see two men load little cartridges on the end of an iron rod with bituminous coal, and push them into the furnace tuyères, and with a piston inject 2 or 3 lb. of bituminous

coal into the smelting zone of the furnace. Moreover, the statement was made to us that without the extra heat generated directly in the smelting zone by this bituminous coal, the furnace did not operate well.

In the further description, at the Ashio works, it is stated that copper is deposited from the mine waters by means of iron. When we visited at Ashio in 1911 the methods had been changed, since the solution flowing from the precipitating plant was destroying the rice fields in the valley. A method of treatment by fractional precipitation by lime was then in use, which I described in a small paper in the *Institute Transactions* for 1912.

At the refining works at Nikko there was an interesting refining plant on the series system, which had been put up by a Japanese who came to this country, and worked at Baltimore, and carried the process *en bloc* to Nikko.

The tradition, however, was faint and the practice was poor. I have heard within the last week that a new plant is being erected to replace this inadequate one at Nikko, using the series system.

Coal-Dust Fired Reverberatory Furnaces

Discussion of the papers of DAVID H. BROWNE, LOUIS V. BENDER, and R. E. H. POMEROY, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 97, January, 1915, pp. 49 to 60, 73 to 81, and No. 98, February, 1915, pp. 445 to 453, respectively.

E. P. MATHEWSON, Anaconda, Mont.—After hearing about the success of D. H. Browne with his furnaces, we in Anaconda decided we might venture into the field of pulverized coal for reverberatory smelting. We sent a delegation of investigators to see Mr. Browne and his work, and they returned, filled with enthusiasm, and said they would recommend it strongly. We thereupon asked for an appropriation, and after some delay obtained the same and fitted up one furnace to use pulverized coal.

We copied Mr. Browne's practice and his selection of machinery for drying and pulverizing coal; and we also copied the burner he used, which was a standard article on the market. We got an expert coal-dust machinery man to design our plant, and were very glad we did so as we avoided some of the unpleasantness that had been experienced at other plants through fires and minor explosions.

I want to say, before describing our apparatus a little in detail, that we were quite familiar with the work of Mr. Sörensen, the pioneer in the use of pulverized coal for copper smelting. Our attention had been called to it by Mr. Klepetko early in the game. And afterward we were

familiar with the work done at Cananea by Mr. Shelby. We sent an expert to investigate the question 10 years ago. He examined the work of Mr. Sørensen; and then went to the cement men and examined their work, and reported at great length that coal dust was the ideal fuel for them, as they took the ash from the coal and sold it as cement. He reported that coal-dust firing could not be made economical in reverberatory copper smelting, and his reasons were that a blanket of non-conducting material would cover the charge in a short time and insufficient heat would reach the charge proper, and we would have floaters over all our furnaces. The flues would be stopped by the dust that did not settle on the charge. We accepted that until our Irish friend got busy and showed us how to do it. Then we built a furnace which we thought was a slight improvement over Mr. Browne's. We put railroad tracks over our furnaces to draw the charge to the furnace. We had long hoppers both sides of the furnace, nearly the full length, and we had pipes leading from these to the sides of the furnace through which we dropped the charge and filled the furnaces almost to the skewbacks. At the start we tried to tap our matte on the side of the furnace. But we found that the furnace seemed to bridge on the bottom and that the matte would not pass the obstruction. Then we had to stop charging in front of the matte taphole; then it struck us that it would be better to tap the matte at the front of the furnace where the slag was drawn off.

This we did, and we put an ordinary tap plate in front, 18 in. below the regular slag outlet. Then we tried to let the slag flow continuously from the furnace, tapping the matte as it accumulated in the sump in front of the furnace.

This worked very nicely. We had once in 8 hr. to take down the front door and skim off the accumulation of unfused ash which might be on the front of the furnace. This was but a few pounds and caused us no inconvenience.

We were so pleased with it that we determined at once to build a bigger furnace. We had our engineers examine the building to see what could be done. We thought we had all we could get in the building; but we learned we could lengthen the furnace a little and put the coal-conveying apparatus outside of the present building. We extended the furnace so that the hearth was 144 ft. in length, and the furnace was widened to 25 ft., inside measurement. This widening of the furnace was not a novelty, as our friends at El Paso with oil firing had reached 24 ft. But 25 ft. was the largest we could get without reconstructing our building. This furnace was put in operation early this month. We put a slow wood fire in the furnace Feb. 5, 1915. This was continued for four days; and then we put in a little coal fire, and on Feb. 9, coaled up to a tonnage of 51 tons of coal for heating this furnace. On the 10th, the day I left there, the furnace was in splendid condition, very

hot, so I instructed those in charge to begin charging the furnace with calcine. They did so, as is evident from the report.

I have a telegram dated the 15th which gives the result:

"On the 10th, 345 tons were charged, and the fuel ratio was 7.52. On the 11th, 517 tons were charged, and the fuel ratio was 7.91. On the 12th, 555 tons were charged, and the fuel ratio was 7.10. On the 13th, 605 tons were charged, the fuel ratio being 7.71. On the 14th, 613 tons were charged, and the fuel ratio was 7.1." And the superintendent remarks, "Everything O. K. for full tonnage soon."

Is any one here familiar with the work done at the Steptoe Valley plant, McGill, Nev? They have a large furnace, the performance of which is mentioned, I believe, in Mr. Pomeroy's paper. I saw that furnace in operation a couple of years ago, and it was doing some remarkable work for an oil-fired furnace.

Performance of the furnace for seven years is given on p. 450, at the bottom of the page. Total charge per furnace, 721 tons; solid charge, that is calcine without the slag, 628 tons. The balance of the charge is hot molten slag.

So they have reached a very good tonnage at that large furnace in Steptoe. Coal-fired furnaces will put through greater tonnages and will give a better fuel ratio than the oil-fired furnaces. That has been proved conclusively at Garfield. At Garfield they are abandoning oil fuel and substituting pulverized fuel throughout the plant. The price of oil, however, has something to do with it, as the oil at McGill is a little cheaper than the oil at Garfield.

CHARLES W. GOODALE, Butte, Mont.—Apparently, the difference between success and failure in this matter of coal-dust firing rests on the degree of pulverization of the coal.

- In 1906 or 1908, at the time of the first experiments in Utah and Cananea, in reverberatory smelting with pulverized coal, I understand the coal was not sufficiently fine to make the combustion complete in the furnaces, and as a consequence more or less half-consumed material fell on the charge, and that was the blanket which they referred to which interfered with the full effect of the heat on the charge itself.

I would like to know from Mr. Mathewson or Mr. Browne if that is not the history of the first work, and the reason why it did not succeed at first.

DAVID H. BROWNE.—I cannot give the exact figures, as I have forgotten them. But I remember that at the Cananea furnace they were using an Aero pulverizer, into which the coal was fed wet and was driven into the furnace by the pulverizer itself. The coal was pulverized, and was dried by the heat of friction, and was blown in, all by the one machine.

Particles of coal 0.1 in. in diameter were not uncommon, and these particles, while they burned, gave a considerable chunk of ash.

I should say that for this work almost all the coal should pass a 200-mesh screen in order to get complete combustion; and what is of more importance, to get a fineness of ash which will be carried out of the furnace by the draft. I do not suppose 5 per cent. of the ash in coal-dust fired reverberatories would stay in the furnace if the coal was pulverized to 200 mesh. In our furnaces we take out two or three bushels of ash at the front of the furnace each 24 hr. I do not suppose we take out over 400 or 500 lb. of ash in a day, if that much.

N. M. LANGDON, Mancelona, Mich.—Although I did not get to Cananea until after the experiments with coal-dust firing were over, I saw the apparatus at work previously, on boiler firing; and what Mr. Browne has said about the lack of fineness of pulverization is perfectly correct. Particles larger than 0.1 in. in diameter were given by this combined breaking and blowing apparatus, which took the lump coal and broke it, and by a crude air-blast forced it in.

JAMES B. HERRESHOFF, New York, N. Y.—The description of coal-dust firing at Anaconda, the new method of charging on the sides, instead of at the center, the life of the bricks, and the size of the units, make very interesting reading.

I feel that the new method of charging reverberatory smelting furnaces is the most important improvement made in several years. The wear and tear on the furnace by this method of charging is now confined to the roof, as the sides and hearth are fully protected by the charge.

In coal-dust firing it may develop that one of the great advantages will be that fuel which cannot be burned upon the grate can be burned after being pulverized. A great deal of coal contains a high percentage of sulphur and iron, which forms clinkers. Of course, this is objectionable in the ordinary method of firing on account of the production of blowholes in the fire, which make the percentage of CO_2 in the gas low. The high efficiency of coal-dust firing is of course due to the small amount of excess air necessary to effect complete combustion.

The long life of the bricks probably results from several causes. The bricks at Anaconda are very refractory, containing a high percentage of silica—if I remember rightly, about 96 per cent.

And then again, the method of charging results in piling a good deal of material on the sides of the furnace, which acts as a flywheel, so to speak, preventing the temperature from rising too high.

Then again, being high in the back, on the firing end, the flame in the furnace does not come into contact with the roof.

They do things in a big way out West. At Anaconda they now

have the largest reverberatory furnaces in the world. The furnaces are five or six times larger than any in the steel industry, and the blast furnaces are three or four times as large; and you can say the same thing about the converters.

At Great Falls, the open-hearth gas furnaces are considerably larger than those found elsewhere. And the stack at Great Falls is much larger than any other in the world.

I think it would be interesting to some of you to know that at one of the large Eastern copper refineries a large furnace has been installed for melting up cathodes. This furnace is 40 ft. long on the hearth; 19 ft. wide, and 7 ft. high, inside dimensions. The waste-heat boiler contains 11,500 sq. ft. of heating surface; that is about 1,150 boiler horsepower. The casting machine is capable of turning out 90 tons of wire-bar per hour. This furnace has already turned out 350 tons of copper in 24 hr., and it is expected that ultimately it will turn out 400 or 450 tons in 24 hr. A furnace, 30 by 16, has turned out 330 tons in 24 hr.

MR. MATHEWSON.—There are many smelting furnaces of large size here running 365 days in the year, and within 30 miles of this spot there is more copper handled than any place on earth.

In connection with the remarks on reverberatory furnaces I neglected to say there was one particular point of economy in the new method adopted for charging the furnaces, and that was the delays for fettling the furnaces were eliminated altogether, saving 10 per cent. time. The charge protects the sides.

W. McA. JOHNSON, New York, N. Y.—I think it would be a good thing to discuss the thermal efficiency of coal-dust firing of copper calcine.

J. W. RICHARDS.—There are not quite enough data in the paper, and I have not had opportunity to cast up the thermal efficiencies of the operations. But if we compare the reverberatory furnace in general with shaft furnaces in general, we find the shaft furnace has small losses by radiation and from the temperature of escaping gases, but a large thermal loss by reason of the percentage of unburned CO in the gas.

Comparing the reverberatory with the blast furnace, the reverberatory is at a disadvantage because of large radiation surface and radiation losses. Also, it works at high temperatures of the issuing gases. Its great advantage is perfect combustion of the fuel with the small excess of air. That is a strong point of the reverberatory furnace and enables it to make its good showing against the blast furnace.

Comparing coal-dust firing with coal-grate firing, the difference is, that in burning coal on a grate not more than two-thirds or three-quarters of the heat generated by the coal by combustion goes into the

furnace. There is a loss of 25 to 33 per cent. of the calorific power of the coal, by radiation from the sides.

By coal-dust firing that is obviated. That is the essential basis of the greater efficiency of the coal-dust firing.

The other improvement which is made, namely, that of charging from the side so as to increase the heat-absorbing surface, is a very keen recognition of the fact that the heating in the reverberatory furnace is almost entirely by radiation from the flame; and therefore, if you increase the angle which the charge subtends against the flame, you increase almost in that ratio the capability of the heat absorption. In the one case you have a flat charge on the bed of the furnace; but banking it up on the sides you increase by one-half or more the area of heat-absorbing surface, and, therefore, the heat which can be put into the cold charge.

The two kinds of firing, the grate firing and coal-dust firing, are alike in this: They both give the total calorific power of the coal, as combustion is perfect; but in the one case you lose one-quarter to one-third of the calorific value of the coal by radiation.

MR. JOHNSON.—I think if Professor Richards could give the meeting a few approximate figures, we would have some way of gauging the relative thermal efficiencies.

MR. RICHARDS.—Roughly, it amounts to this: If you lose one-quarter of the calorific value of the coal in a fireplace, you have only three-quarters to be utilized in the furnace. The heat loss at the flue end is a constant and would be the same for both.

Let us say the charge is heaped the same way on both furnaces, the amount of heat then absorbed would be the same in both, and in one case you would have some ratio of 75 per cent. and in the other some ratio of 100 per cent.

If in grate firing you have 15 per cent., you have 15 per cent. of 75 per cent., or about 11 per cent. In the other case you get 15 per cent. of 100, or 15 per cent. net.

MR. BROWNE.—I want to say that most of the credit for this process is due to the boys on the spot. The best we can do is to give the boys an idea, and they make a success of it.

Almost one-half of the gain we have made in coal-dust firing is due to the way the charge is put in. Neither Mr. Sørensen nor Mr. Shelby used that. They both used the side system for fettling only. One of our foremen, now superintendent, began to use ore for fettling instead of quartz. That was so satisfactory that we discarded the charging hoppers and substituted fettling through these fettling holes. That was brought out and worked out entirely by this smelter foreman. He deserves the credit for it.

MR. MATHEWSON.—I think I can help Mr. Johnson to a solution of the problem about which he asks. I can say that in the Anaconda practice we have gradually worked things down so that the combined briquet and blast-furnace expense was about equal to the combined roasting and reverberatory expense. Then on the introduction of the coal-dust fired reverberatory furnace we cut off 50c. a ton of expense, and that is probably the economy in fuel. That 50c. will mean that instead of operating blast furnaces as well as reverberatory furnaces at Anaconda, we will put our blast furnaces to one side, and concentrate all our ore with jigs, roughing tables and an improved flotation process. All our smelting will be done in coal-dust fired reverberatory furnaces.

Then, to handle the by-products, we have in mind the building of a reverberatory furnace 175 ft. in length by 25 ft. in width, in which we will put all the molten converter slag and flux same with siliceous concentrates from the flotation process. These will carry sufficient sulphur to clean the copper out of the slag, and silica to bring the slag up to the normal composition of reverberatory slag, which can be thrown away.

We have figured that if we leave as much as 0.6 per cent. of copper in this special reverberatory slag we will beat the blast furnaces.

HENRY D. HIBBARD, Plainfield, N. J.—I notice that some of the roofs are 20 in. thick. Are the bricks 20 in. long?

MR. MATHEWSON.—Twenty inches long.

FRANK KLEPETKO, New York, N. Y.—I do not think you can ascribe all the benefit of the improvement in the quantity of material smelted and in the greater ratio of tonnage of ore to coal smelted to the pulverizing of the coal. I believe it is largely due to the charging on the side and firing on the charge instead of on the slag; and if some one had widened the furnace and built it higher so as to get the charge higher on the side, they might have improved the grate-fire practice.

MR. RICHARDS.—You cannot get away from the fact that at least 25 per cent. of all the calorific power of the coal is lost in the fireplace by heat transmission through the walls of the fireplace, and that the gases coming in carry only 75 per cent. of it. That is put in by the coal-dust firing, and the improvement.

MR. HIBBARD.—But a man might not have the money to install his apparatus for pulverizing the fuel. He might have, and if he adopted the method of firing on the charge instead of on the furnace he would get different results. The doubling up of the ratio is not entirely due to the pulverizing of the fuel.

MR. MATHEWSON.—That was brought out by Mr. Browne. Mr. Browne called attention to the fact that the proper way to fire was to

put the fire in the center of the charge, and have the charge all around the fire; and we have got as near to it as we can with the fixed furnace.

ALBERT F. SCHNEIDER, Plainfield, N. J.—How large a tonnage do you expect in your new big furnace?

MR. MATHEWSON.—We expect to get 750 tons through the furnaces daily, with a ratio of 7.5 to 1.

F. C. NEWTON, Maurer, N. J.—I was interested in what Mr. Browne said about the amount of dust or ash left in the furnace. That problem has come up in the casting of copper; and one argument against the use of coal-dust firing in casting anodes or casting wirebars was the possibility of them dropping in and being slagged by the copper oxide, and requiring more copper to be in circulation back of the blast furnace. That is a very interesting point. If only 5 per cent. of the ash falls on the bath that is practically negligible in the casting of copper.

MR. BROWNE.—I do not want to be understood as limiting this to 5 per cent. I said two or three barrows of ash a day (about 500 or 600 lb.) are raked out. This ash is like pumice stone; it sticks a little at the throat of the furnace, where we have a rake-out door to clean out this accumulation.

MR. NEWTON.—In the anode furnace the method is to charge it by filling it with as much copper as the charging machines can handle, and place it in the furnace. I was wondering whether that would increase the amount of ash in the furnace, retarding the gases going through.

MR. BROWNE.—I do not think so.

MR. SCHNEIDER.—In changing from the blast furnace to the reverberatory, do you expect to lose more or less copper?

MR. MATHEWSON.—We expect the copper loss to be the same. The blast-furnace slags run lower in copper than those from the reverberatory furnaces, by about 0.1 per cent. of the total weight of slag. But the amount of lime added to the charge counterbalances that. The lime contains no copper.

H. S. MUNROE, Concrete, Colo. (communication to the Secretary*).—A few remarks on Western cement practice in this matter may be acceptable.

In two cement plants operating on the same coal, which I have had occasion to observe more or less closely, the practice varies rather widely. In each of these plants Walsenburg slack is used, which obviates the necessity of using rolls.

In the first plant considered, the coal is dumped into a receiving

* Received Jan. 5, 1915.

hopper and elevated into two cone-bottom bins, from the bottoms of which it is drawn by two screw conveyors, delivering to a common drag conveyor. The drag passes the coal over a magnet to eliminate the iron (a very important factor, as will be shown) and the coal is fed by gravity into a Ruggles-Coles drier.

At the discharge end of the drier the coal is dumped into the receiving hopper of a No. 6 Universal Williams hammer mill, having $\frac{3}{32}$ -in. grate-bar openings. The hammer mill discharges into an elevator pit, from which the coal is elevated into a bin which feeds a No. 16 Smidth tube mill, in which $\frac{3}{4}$ to $1\frac{1}{4}$ in. nut punchings are used as pebbles. This mill discharges into another elevator, which in turn discharges into a screw, taking the coal to the kiln coal tanks.

Suitable dust-chamber provision is made for dust from the drier, the discharge of which leads into the elevator pit preceding the tube mill.

This mill has a capacity of 100 tons per 20-hr. day, and the resultant product has an average fineness of 96 per cent. through 100 mesh. The power consumption, including stock piling, done by overhead screw conveyor and reclaiming from stock pile done by three longitudinal screw conveyors, each 96 ft. long, and one transverse screw conveyor 40 ft. long, is about 30 kw. per ton, and the total cost over any year, including all maintenance charges, is about 35c. per ton of coal ground.

With a greater finish-grinding capacity, the hammer mill could be eliminated on slack coal and a power economy be effected.

The addition of rolls before the drier would adapt a plant of this type to mine-run or lump coal.

The other plant mentioned has a capacity of about 200 tons per day on the same material. In this the coal is dried in a rotary drier of local manufacture and, without any preliminary grinding, is fed to a 6-ft. Griffin mill and a No. 16 tube mill, both of which turn out a product of 95 per cent. average fineness through 100 mesh.

I have not had access to cost figures on this plant, but think 27c. per ton would amply cover.

On the construction of coal-grinding plants, several salient points should be borne in mind.

1. There should be *no* wood anywhere in the construction. A coal-grinding plant is on fire somewhere at least 50 per cent. of the time it is operating, especially with an ordinarily gassy coal. This fire will be present wherever coal dust is allowed to accumulate.

2. Ample ventilation should be provided, to prevent the accumulation of a dust-laden atmosphere in the top of the building, a condition inviting explosion.

3. Power lines, where possible, should be run under or around plant, but when necessary to run inside plant, should be in conduit. Only

incased lights should be used. Motors should be kept outside plant where possible, and when inside, should be only induction motors. A spark from the commutator of a direct-current motor will often provide the necessary ignition to set off the whole works. All electrical controls should be outside the possible dust zone.

4. Driers should be designed to be fired outside the plant.

5. Iron should be surely eliminated from the coal where hammer mills are used; 95 per cent. of the explosions which have occurred in the plant first considered have been occasioned by iron being heated to incandescence in the hammer mill, causing a general explosion which invariably cleaned out the building of combustibles.

There is no particular danger in the operation, when every possible precaution is exercised. I know of one plant where nine men were burned to death while cleaning down the upper part of the mill, but this dust could not have been ignited without fire. On the other hand, another plant has operated eight years without seriously burning any one, though numerous explosions have occurred. No doubt there are many plants with as good records.

W. D. LEONARD, Garfield, Utah (communication to the Secretary).—The following are some of the determining factors leading up to the investigation and final adoption of coal-dust firing for copper-smelting reverberatory furnaces at the Garfield plant.

1. Great difference between the cost of fuel oil and the cost of its equivalent heat units from coal was one of the deciding factors.

The thermal heat value of the ordinary coal is about 12,200 B.t.u. per pound. The thermal heat value of California oil is about 18,400 B.t.u. per pound.

About $1\frac{1}{2}$ tons of coal are required to equal in heat value 1 ton of oil, and when the price of coal is considered it will be found that the ratio in favor of the coal is about 3 to 1.

2. The belief and final confirmation was reached that powdered coal could be burned nearly as efficiently as oil; or to put it more clearly, that the heat units required to smelt our charge would be about the same whether oil or powdered coal was used.

The essential part of the equipment required for powdering coal is, as follows:

(This description of the equipment has been made as brief as possible inasmuch as Mr. Bender's paper covers the equipment in detail and our equipment and that of Anaconda are identical, with the exception of the size of drier.)

Our drier is the Ruggles-Cole No. A-8, 30 by 5 ft., rated at 10 tons per hour. We have not been able to run the drier to full capacity as our furnace and storage are not equal to full capacity of the drier. We have

averaged 7.5 tons per hour; water in coal before drying, 3.7 per cent.; after drying, 1.5 per cent., and requires 1.42 per cent. of coal for drying. One Raymond 5-roller pulverizing mill rated at 5 tons per hour, together with accessory fans, conveyors, and separators. We have averaged 4.1 tons per hour and so far the mill has been satisfactory. Four Sturtevant coal burners.

The plant was installed for our No. 6 reverberatory and put into operation about May 1, 1914. During the first ten days of operation considerable delay was experienced, due to minor defects, but the plant has operated very satisfactorily since.

Tests were made on various Utah and Wyoming coals and it was found in general that bituminous coals were superior to the lignites, all things being considered.

As regards the use of powdered coal in the furnace, there is very little difference to be noticed from oil firing. No difficulty has been encountered from ash accumulations in flues or boilers, or blanketing of charge. The furnace arches over the combustion chamber apparently last as well as in oil firing. The fettling, however, is smelted out near the fire end and at the bridge wall rather faster than in the case of oil.

It has been thoroughly demonstrated here that for the average use of our coal, about 10 per cent. more heat units are required to smelt a ton of charge than are required in oil firing. This was arrived at by comparing the final ratios over periods of 30 days, and finally over the whole period of operation.

This difference was not constant, however. In one case, using a short-flame lignite, the ratio of heat capacities of oil and coal was 75 to 100, and in our best run on bituminous coal, 99 to 100.

Fuel ratios have shown a wide variation, owing to the use of different coals in the furnace, also conditions and character of charge smelted (note log of furnace).

Furnace capacity is largely a function of the amount of fuel burned, or that can be burned, in a given furnace. In operating our furnace, we forced the tonnage up to 460 tons per day and could have exceeded this figure if we had so desired. But our coal ratio showed a tendency to increase, and further, we were afraid the wear and tear on the brick work was not in an equal proportion to the tonnage smelted, and therefore discontinued the practice. It was considered that about 400 tons per 24 hr. was the most economical basis for our furnace and conditions.

The control of the fire or flame in a furnace using oil or powdered coal is important with respect to fuel economy. In oil firing at Garfield we aim to clear the flame at the last door, or about 30 ft. from the skimming door. At times of charging, the flame sweeps to the throat and gradually shortens until it is barely visible at this point. Draft at this point is about 0.15 in. of water. Under this practice, determinations have

shown that the gases escaping at the throat average about 2.5 per cent. free oxygen, which is equivalent to about 12.1 per cent. excess air. At Garfield we consider this good practice.

In operating with powdered coal, we use a slightly longer flame, but do not permit it to reach the boilers. When clearing the flame, as in oil firing, we have obtained an average of 5 per cent. free oxygen, which corresponds to nearly 25 per cent. excess air.

Efficiency of Coal as Compared to Oil

The oil, being an ideal fuel and very uniform in composition, heat value, etc., is used below as a standard in making comparisons. In making the tests, the powdered coal was compared to oil firing during the same period on the other furnaces, all of which were smelting the same charge. Conditions in each case were therefore as nearly as possible identical.

The ratio of the heat units in the oil and coal used per ton of charge smelted is arbitrarily taken as an expression of efficiency.

Name	Kind	Coal, B.t.u. per Ton Charge	Oil, B.t.u. per Ton Charge	Efficiency, Per Cent.
Castle Gate.....	Slack B	4,590,000	3,956,000	86
Rock Springs.....	Slack L-B	4,210,000	4,011,000	95
Black Hawk.....	Fines B	5,134,000	4,673,000	91
Standard.....	Dust B	4,795,000	4,673,000	97
Bamberger.....	Slack L	5,498,000	4,011,000	73
Black Hawk.....	Dust B	4,451,000	3,772,000	84
Mohrland.....	Slack B	4,715,000	4,011,000	85
Independent.....	Slack B	3,785,000	3,772,000	99
	Averages	4,647,000	4,110,000	88

In no case has it been observed that the efficiency as considered above has exceeded that of oil.

No well-defined reasons have yet been advanced for these variations of efficiency.

In the second column, coals are marked "B" and "L" to designate whether the coal is bituminous, lignite, or intermediate.

Metallurgical results have not differed from those obtained in oil firing.

Our No. 6 reverberatory furnace is 19 by 112 ft. with arch 7 ft. 10 in. from the floor level, and is fed by two rows of hoppers 6 ft. 6 in. from the side walls, thus leaving 6 ft. of space between the hoppers in the center of the furnace.

Log of No. 6 Reverberatory Furnace, Garfield Plant. Powdered Coal Firing—July 22 to Sept. 1, 1914

Date, 1914	Charge, Tons	Coal, Tons	Coal, Per Cent. Charge	Drying Coal, Lb.	Powdered Coal			Name	Oil on Other Furnaces	Remarks
					Ash Per Cent.	H ₂ O Per Cent.	Through 200-mesh, Per Cent.			
7-22	396	30	20.2	1,900	7.0	1.45	71.5	Black Hawk...	0.66	247 tons by oil..
23	439	56	15.0	1,900	7.1	1.40	71.7	Black Hawk...	0.63	54 tons by oil.
24	383	60	15.7	2,500	7.3	0.90	72.5	Black Hawk...	0.59	
25	385	70	18.2	2,000				Black Hawk...	0.63	Clayed.
26	343	68	19.8	2,000	7.8	1.30	73.8	Black Hawk...	0.60	
27	347	70	20.2	1,300	9.0	1.50	75.6	Black Hawk...	0.65	
28	338	70	20.7	400	9.7	1.60	74.5	Mohrland.....	0.63	
29	290	48	17.9	2,900	7.8	0.30	71.5	Mohrland.....	0.67	22 tons by oil; clayed.
30	360	69	19.2	2,300	7.6	0.60	72.4	Mohrland.....	0.69	
31	415	77	18.6	2,200	7.9	1.30	67.7	Mohrland.....	0.74	
8-1	370	70	18.9	2,200	7.8	1.15		Mohrland.....	0.71	
2	392	68	17.4	1,800	8.0	1.15		Mohrland.....	0.62	Clayed.
3	335	74	22.1	400		1.00		Mohrland.....	0.59	
4	312	65	20.8	2,000	7.5	0.85		Mohrland.....	0.63	
5	414	68	16.4	2,000	7.8	0.76		Independent...	0.60	
6	408	62	15.2	1,600	6.5	0.93		Independent...	0.62	
7	397	64	16.5	1,100	7.4	1.00	75.6	Independent...	0.63	9 tons by oil; clayed.
8	415	60	14.5	1,400	7.4	1.63	67.9	Independent...	0.63	
9	417	55	13.2	1,900	8.1	1.71	70.0	Independent...	0.63	
10	379	55	14.5	1,800	7.7	1.35	71.1	Independent...	0.63	
11	401	56	13.7	1,800	7.4	1.35	71.1	Independent...	0.60	Clayed.
12	401	56	13.7	1,900	6.9	1.52	73.3	Independent...	0.60	
13	316	48	15.3	1,100	6.5	1.44	72.3	Independent...	0.63	
14	345	55	16.0	1,700	6.8	1.49	70.9	Independent...	0.64	
15	413	60	14.8	1,900	6.4	1.27		Independent...	0.64	Clayed patch.
16	353	60	17.0	2,200	7.0	2.34	69.3	Independent...	0.64	Clayed.
17	370	60	16.3	1,900	6.2	1.33	72.5	Independent...	0.64	
18	376	55	14.6	600	6.8	1.71	70.7	Independent...	0.63	
19	370	55	14.8	2,000		1.54		Independent...	0.60	Bridge clayed.
20	414	56	13.5	2,200	7.8	1.15	69.7	Independent...	0.62	
21	378	56	14.8	2,000	6.2	0.94		Independent...	0.61	Clayed patch.
22	370	58	15.3	2,200	7.8	2.18		Mohrland and Independent.	0.60	
23	399	60	15.1	600	6.7	2.21		Black Hawk...	0.55	Clayed.
24	440	56	12.7	1,900	8.5	1.65		Hiawatha.....	0.62	
25	418	55	13.2	2,000	7.1	1.38	70.2	Independent...	0.62	
26	407	60	14.8	2,000	8.0	2.25	73.8	Independent...	0.61	Clayed.
27	389	62	15.9	2,000	6.0	3.80	72.8	Black Hawk...	0.67	Did not dry.
28	373	55	14.7	1,500	6.8	1.94	71.9	Black Hawk...	0.56	
29	327	55	16.9	1,600	7.2	1.90	72.0	Mohrland.....	0.70	Bridge clayed.
30	330	55	16.6	1,500	8.5	2.32	74.5	Mohrland.....	0.65	
31	Furnace shut down for repairs.									
Avg.	378	60	15.8	1,755	7.4	1.47	71.9		0.63	

Average per cent. coal through 200 mesh, Sept. 10 to Oct. 1, 77.1.

The Limits of Mining Under Heavy Wash

Discussion of the paper of DOUGLAS BUNTING, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 97, January, 1915, pp. 1 to 21.

ARTHUR HOVEY STORRS, Scranton, Pa.—I know something about accidents Nos. 4 and 8. At the Fuller mine there was no disturbance of the surface in connection with either of those accidents, there was no surface settling, nor signs of sand in the mine, and seemingly the water all came from a subterranean basin or water-bearing stratum overlying the Six Foot vein.

A similar thing was shown, in an accident not referred to in Mr. Bunting's table, at the Avondale, where we had an inflow of water, as high as 12,000 gal. a minute, for a considerable period.

Regarding the second accident (No. 8) at the Fuller mine: after the first inflow of water (No. 4 of the table), the mining was abandoned in the Six Foot vein. A long slope was driven in an underlying seam, and when this had reached a point about 2,000 ft. southeast of the place where the first inflow of water occurred, and where the bore holes show there was apparently sufficient rock covering for the Six Foot vein, a rock plane was driven up to the upper vein (Six Foot) and one afternoon this exposed the coal. During the following night the inflow of the water occurred. There were absolutely no workings at this point so that no disturbance of the overlying rocks could have occurred.

The result of the inflow of water was to fill up the entire lower-vein workings in about two or three days and caused the abandonment of the mine.

After several years, considerable time and money were spent in an effort to hoist the water from the shaft and mine workings, with varying, but in the main rather indifferent, success until, due to the breaking of one hoisting rope, the engineer pulled the other hoisting tank up through the tower and down on to the engine, making a complete wreck of the whole plant. It was then decided to allow the mine to soak and it is still doing so, other portions of the property being mined from adjoining collieries.

Later came the big inflow that I spoke of, the 12,000 gal. per minute at the Avondale. In this case there was much disturbance of the surface; the depression showing from 4 to 6 ft. over an area of several acres. The surface covering varied from 6 to 80 ft. of gravel, sand, and clay, with the Susquehanna River flowing over same not far distant. It was generally contended that the water came from the river through these surface materials, but soon after the inflow a rise of the river put about 6 ft. of water over the sunken area without showing any increase of

the inflow into the mine. The heavy original rate of inflow continued for some weeks and then gradually decreased until it reached a minimum of about 4,000 gal. a minute, at which rate it would flow for months. Then after a long rainy period the inflow of water would gradually increase in volume. Apparently there was a subterranean basin of water which, standing full and being tapped by the mine caving, caused the original heavy rate of inflow. This gradually drained down until the minimum inflow represented the ordinary normal amount of ground water made in this basin. Heavy continued rains gradually increasing the amount of ground water, a higher rate of inflow would be maintained for a period. The amount of water over, or in, the immediately overlying surface gravel and sand and the height of the nearby river did not apparently affect the rate of inflow.

Similar conditions to those of this Avondale inflow apparently have existed at other places in the Wyoming Valley where inflows of water have occurred. It would appear that perhaps there was a large subterranean water stratum from which these water inflows came, rather than from the surface gravel and sand. The analysis of the inflowing water at the Avondale was markedly different from that of the river water.

CHARLES ENZIAN, Wilkes-Barre, Pa.—I desire to offer a suggestion in connection with accident No. 3, which occurred in a mine adjoining the one Mr. Storrs has spoken of. The mining engineers of the Wyoming Valley have for many years suspected the existence of a subterranean water channel in the vicinity under discussion. I think substantial proof of its existence may be taken from the experience in the construction of domestic water-supply wells. In the bottom of the upper water-bearing strata there is a clay bed of variable thickness. If the wells are sunk only to the clay they will retain the water and yield a good supply, but if the clay bed is pierced the water is lost.

Following the occurrence of accident No. 3, and directly over it, an inverted cone-shaped surface depression existed, measuring about 175 ft. in diameter at the surface and said to have been originally from 20 to 30 ft. deep. It is possible that in the accident to which Mr. Storrs refers, the water in the sub-channel was tapped in such manner as not to draw in the entire sand strata; whereas the break in the No. 3 accident apparently was large enough to draw in the clay bed and extend to the surface.

DOUGLAS BUNTING (communication to the Secretary*).—The suggestion that tests be made of the different kinds of stone overlying the various coal seams is a good one. In the making of such tests, which would be primarily for transverse strength, it would be advisable also to

* Received Mar. 24, 1915.

determine their compressive strengths for various ratios of height to least lateral dimension of specimens.

With reference to the Illinois limestone mine, with longwall coal mine underlying, and the instance where there was an inflow of 12,000 gal. of water per minute into an anthracite mine: In these instances it is probable that the failures were due to insufficient or improper distribution of pillar support, and that being the case, the discussion of these cases would not probably come under the subject of this paper, for reasons given on p. 19 in the second condition for consideration of the use of the formula for determining minimum safe rock thickness.

Relative to accidents Nos. 4 and 8 at the Fuller mine: There is no evidence of disturbance upon the surface now, and there may not have been at the time of the accidents. That being the case, comparatively little sand could have entered the mine, although it has been stated at various times that the inrushes consisted of sand and water. It would, however, be natural to assume that accident No. 8 was accompanied with an inflow of more sand than accident No. 4 and, consequently, more cause for surface disturbance.

The water in the Fuller mine is being removed and possibly, at some future time, we will know something more definite as to the quantity of sand in the mine at the location of these two accidents; also with the lowering of the water it will be proved to what extent the openings which caused these accidents have been closed and what effect the years of standing have had upon their permeability.

Underground Haulage by Storage-Battery Locomotives in the Bunker Hill and Sullivan Mines

Discussion of the paper of J. W. GWINN, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 239 to 247.

GIRARD B. ROSENBLATT, Salt Lake City, Utah (communication to the Secretary*).—The storage-battery locomotive certainly has its place in metal mining and I am glad to note that it is now receiving the recognition that is its due. It is, however, not a panacea for all conditions of mine haulage. There are places where it will not work to advantage and cannot compare with the trolley-type locomotive in cost of operation or in cost of maintenance.

On the one hand, there are places where it clearly outshines the trolley-type locomotive. Much depends on the headroom available in the drifts and on the condition of the roof. For short hauls with moderate-

* Received Jan. 26, 1915.

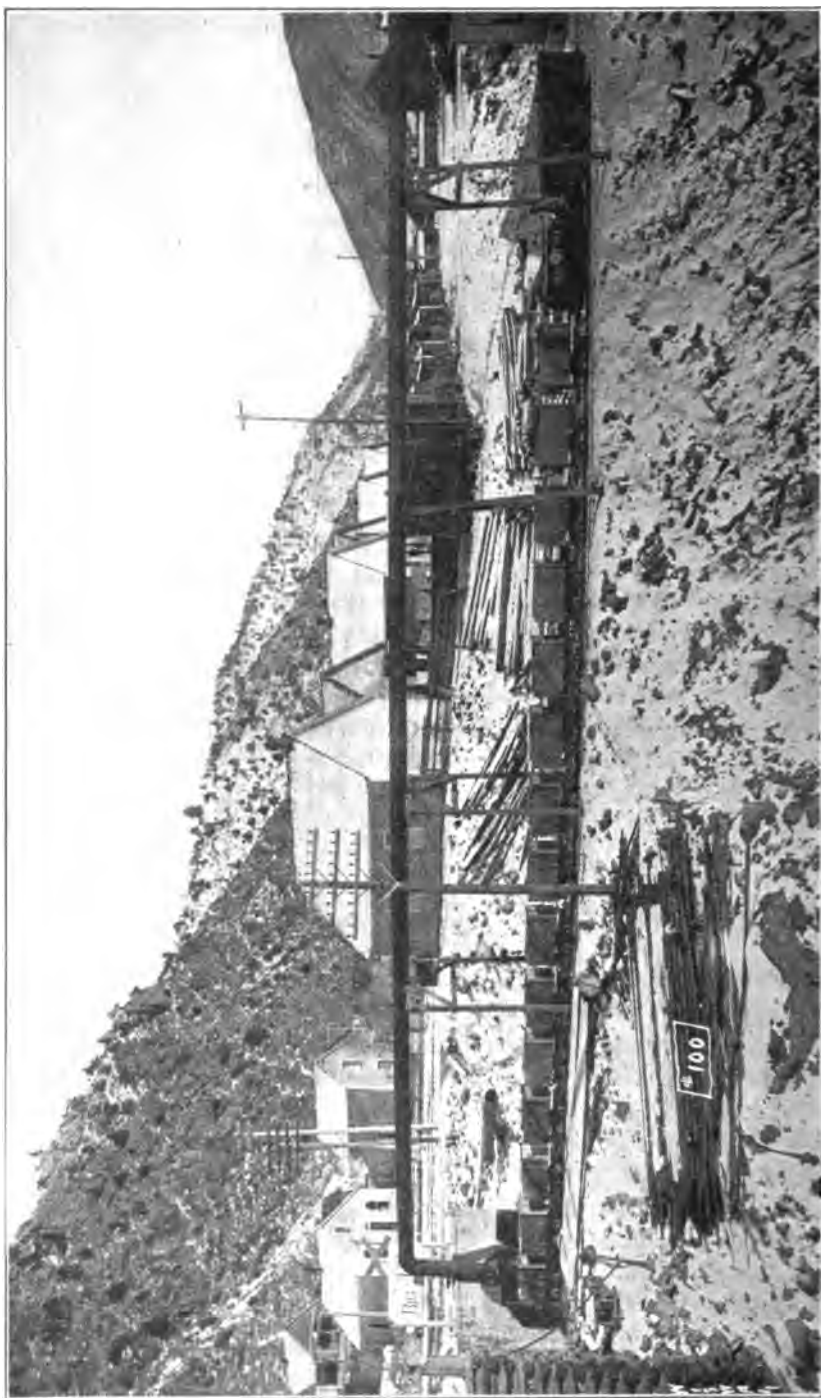


FIG. 4.—HAULAGE AT THE BIG FIVE MINING CO. BATTERY CARRIED ON A TENDER WHICH IS SEEN AHEAD OF THE LOCOMOTIVE.

weight trains, where the roof is low and the ground heavy, nothing can compare with the storage-battery locomotive for convenience, flexibility, and above all, ability to stay on the job without the loss of time required to fix the trolley wire, to replace trolleys, etc. The power economy of a storage-battery locomotive is not as good as that of a trolley locomotive doing the same work. But no mining man cares for a few kilowatt-hours more or less if the use of them can obviate delays in getting out ore.

On the other hand, for long hauls with heavy trains, the storage-battery locomotive cannot compete successfully with the trolley type. The nearest approach to the successful commercial operation of a storage-battery locomotive on long-haul work is that at the Big Five tunnel in Colorado, where 80-ton trains are handled, and in this installation the size of battery required is so large that it cannot be carried on the locomotive, but is mounted on a tender, as shown in Fig. 4.

Records at the Big Five covering a period of six months show a monthly repair charge on the 4-ton Westinghouse storage-battery locomotive with its tender of \$12.60 per month, and a repair charge per ton-mile of approximately \$0.008. It may be interesting to compare these figures with figures that may be forthcoming from other sources. In analyzing these costs, the high cost per ton-mile over the six-month period is due to the fact that for three months the tonnage haul was very light indeed and the figure given above is the average over six months. For two months during which normal tonnage was handled, the repair cost per ton-mile was about \$0.0065, or approximately the same figure as arrived at by Mr. Gwinn for operation on the No. 12 level.

Referring to the installation at the Bunker Hill and Sullivan, it may be of interest to know that the Westinghouse locomotive reported upon is not of a standard storage-battery type, inasmuch as the manufacturer was required to furnish a locomotive suitable for both trolley and storage-battery operation. This necessitated a locomotive with a somewhat different arrangement of controller and wiring than would be required for straight storage-battery operation, and required more complications than are included with the standard storage-battery locomotive. Still the operation of this locomotive has been satisfactory, though its efficiency when operating off the storage battery is not as high as would be the case with a straight storage-battery locomotive. The locomotive does include the special form of control furnished by the Westinghouse company on most of its storage-battery locomotives whereby high draw-bar pulls at slow speeds may be obtained with a less draft of current from the battery than is possible with the usual type of control furnished on trolley-type locomotives. The locomotive with the trolley pole attached is shown in Fig. 5.

It would appear that the average cost of repairs given by Mr. Gwinn in the table on the seventh page of his paper is not a direct indication of

the mechanical cost of operating the locomotives because, as the author states, certain changes in the construction of the locomotives made by the mining company before the locomotives were put into operation seem to be pro-rated over the time of operation. Therefore, we are apt to get the curious result that the longer the locomotive remains in operation the lower will be the monthly repair cost charged against it.

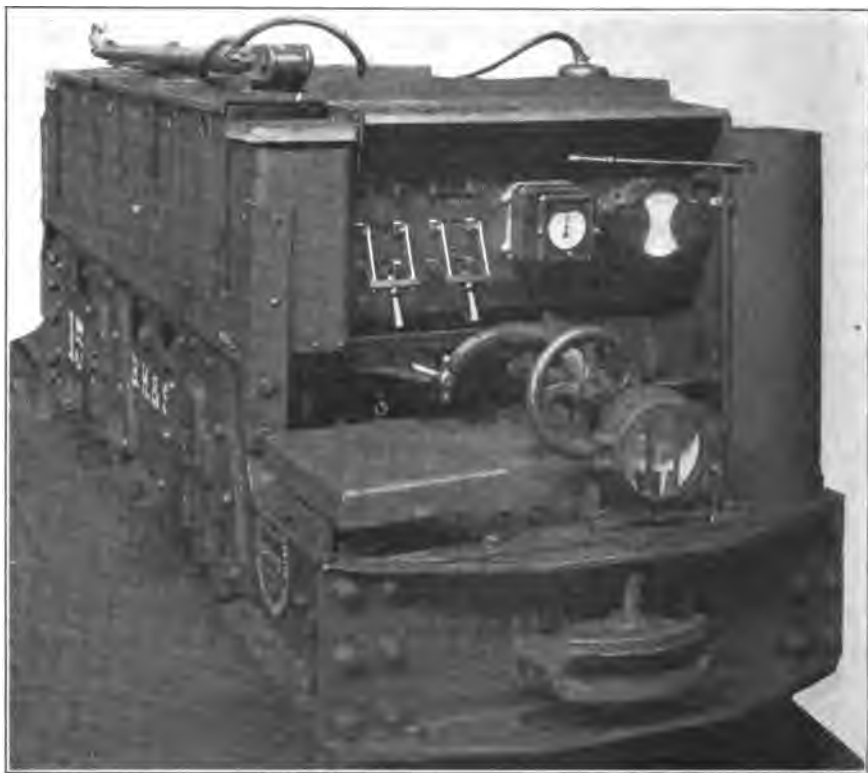


FIG. 5.—WESTINGHOUSE STORAGE-BATTERY LOCOMOTIVE AT BUNKER HILL AND SULLIVAN. NOTE TROLLEY POLE AND CONNECTION. ALSO SWITCH FOR THROWING MOTORS ON TO EITHER TROLLEY OR BATTERY.

It would also seem in comparing the repair costs on the storage-battery locomotives and on the trolley locomotives given by Mr. Gwinn that conditions differing on the various levels have something to do with the repair costs. Combining Mr. Gwinn's two repair-cost tables on storage-battery and trolley locomotives, respectively, into one table gives us the following, which makes it seem that conditions on the No. 12 level are more arduous than on the No. 11 level. It would be interesting to hear from Mr. Gwinn how these conditions actually differ.

	Type of Locomotive	Monthly Repair Charges	Repair Charge per Ton	Reduction in Costs Obtained by Using Storage-Battery Type, Per Cent.
Level No. 11	Storage battery	\$48.513	\$0.00359	17.5
	Trolley	39.88	0.00435	
Level No. 12	Storage battery	55.456	0.00434	32.2
	Trolley	93.912	0.00641	

The higher maintenance cost of the trolley locomotives compared with the storage-battery locomotives is certainly contrary to the usual belief of operators, and it is interesting to have such exact figures based on carefully kept records. The keeping of records which will permit the maintenance cost of locomotives being calculated per ton-mile is certainly a distinct advance in scientific mine management. I do not know of any metal-mining property where such records are available except possibly at the Phelps-Dodge properties in Arizona. Experience at other mines where both trolley and storage-battery locomotives have been operated under approximately similar conditions has led to a belief that the maintenance cost of storage-battery locomotives is higher per ton-mile than similar maintenance costs of trolley locomotives. There have, however, been too few actual records kept for any one to make any general positive statements regarding this matter. The beliefs in vogue are based on definite impressions rather than upon definite facts. At one property in Montana it was found that the principal cost of maintaining a storage-battery locomotive was the necessity of periodic semi-annual overhauling of the storage-battery cells and the battery boxes. A certain mine in Alaska, which operates several storage-battery locomotives, found periodic overhauling and painting of the battery box necessary to maintain continuity of service. Most mines operating storage-battery locomotives have had some trouble in repairing damage to the charging devices, particularly the plugs and plugging receptacles. None of these detailed troubles have been mentioned by Mr. Gwinn, and it would be interesting to hear whether they have been experienced at the Bunker Hill and Sullivan.

That storage-battery locomotive haulage can be thoroughly successful and satisfactory, and is so at the Bunker Hill and Sullivan, is borne out by the fact that affiliated properties in Alaska, the Alaska-Juneau Gold Mining Co. and the Alaska-Treadwell Gold Mining Co., have both installed storage-battery locomotives on the basis of the experience with this type of haulage at the Bunker Hill and Sullivan.

The Alaska-Juneau Gold Mining Co. has within the last year placed in operation one 5½-ton Westinghouse-Baldwin storage-battery locomotive, and the Alaska-Treadwell has in the same period installed two 4½-ton Westinghouse-Baldwin storage-battery locomotives. One of these is illustrated in Fig. 6.

The successful use of storage-battery locomotives in underground workings necessitates the realization that their inherent electrical characteristics are different from those of the trolley locomotive. The

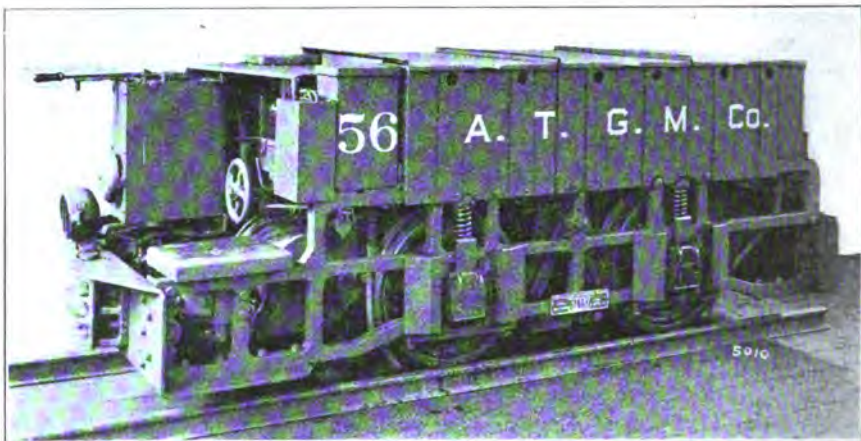


FIG. 6.—4½-TON STORAGE-BATTERY LOCOMOTIVE FOR THE ALASKA GOLD MINING CO. NOTE THE BAR STEEL FRAME.

storage-battery locomotive carries its own power plant with it, and that power plant has a very decided characteristic in that the pressure under which it delivers its power falls off very rapidly when the power is withdrawn at a high rate. In some respects it is like a small steam boiler, the pressure of which would fall off rapidly if too great a draft of steam is required. The operator who would successfully use storage-battery locomotive haulage should bear in mind the following:

Do not try to use too small a battery; a safety factor in battery size will pay for the increased cost of battery within a year.

Do not use motors too big for the battery; the motorman will get all he can out of the locomotive and big motors give him an opportunity to club the battery.

Do not get a locomotive mechanically unsuited for mine service; underground conditions are quite different from surface conditions, and a locomotive to be successful underground must be designed and built as a mining locomotive.

JOHN LANGTON, New York, N. Y.—I would like to call attention to p. 244, where the average kilowatt-hours per ton hauled are given.

They are worked out per ton of ore hauled per 100 ft. Reducing to a ton-mile basis, it comes to 1.79 kw-hr. per ton-mile of ore. Compare the paper read a year ago by C. Legrand on storage-battery locomotives in the mines at Bisbee. In that paper he gave certain figures which I will read, as follows: "The power required at the power station per useful ton-mile was approximately double that required with trolley locomotives, or 1.6 kw-hr.," so we have 1.6 kw-hr. to compare with the figure of 1.79 kw-hr. given in this paper. These are in substantial agreement, and constitute useful data for estimating. But although the amount of power taken, as Mr. Legrand states, is about twice that of the trolley locomotive, still the additional cost of 0.8 to 1.0 kw-hr. per ton-mile is such a small part of the total cost of hauling that it will seldom need to be considered if other advantages gained by the use of storage-battery locomotives should make them preferable to a trolley system.

H. M. CRANKSHAW, Lansford, Pa.—Mr. Gwinn states that the original cost of the storage-battery locomotive is high. We should like to know how much higher it is than the ordinary trolley locomotive. Then another feature which will have to be taken up will be the life of the battery. How long will the battery last, and what is the cost of replacing the battery? It seems to me that depreciation might play a very important part in the monthly upkeep of this locomotive if it were properly reckoned in.

Another point—Is the storage-battery locomotive safe? I do not mean safe as compared to the trolley locomotive, but could it be constructed so as to work in a gaseous place where, for example, mules work to-day? If it could, then it might be a useful help in haulage, otherwise I do not see that it is much of a help.

H. H. CLARK, Pittsburgh, Pa.—The Bureau of Mines is greatly interested in storage-battery locomotives because, as Mr. Gwinn has pointed out, the use of storage-battery locomotives will eliminate the trolley wire, which is about the most dangerous piece of electrical equipment used in mines.

We therefore welcome any report which tends to establish the practicability and economy of storage-battery locomotives.

The impression that I have received from a rather hasty reading of Mr. Gwinn's paper is, however, almost too good to be true. The paper seems to state, without qualifications as to size or character of service, that storage-battery locomotives have advantages over trolley locomotives with respect to economy and efficiency as well as in respect to safety.

While it seems reasonable to believe that storage-battery locomotives can be maintained as cheaply as trolley locomotives, the question arises, Will the operating costs and the charges for interest and depreciation of

the storage-battery locomotive and its charging equipment compare favorably with those of the trolley locomotive? And will this be true for all classes of service; that is, for long hauls and heavy hauls as well as for short hauls and lighter loads? Or is the superiority of the storage-battery locomotive limited to certain kinds and conditions of service?

GEORGE R. WOOD, Philadelphia, Pa.—I can possibly contribute a little practical experience to this discussion. We installed about nine months ago two 7-ton storage-battery locomotives in a coal mine, for reasons of safety. These locomotives are intended to be used on room entries and in pillar work where, under the Pennsylvania mine law, we are not allowed to use trolley locomotives. So far as the State Department of Mines is concerned, they are acceptable, although they would not be considered safe for working in gas. The motors, of course, are inclosed, as ordinary mining motors are, but to no greater extent, but the sparking, if any, would be at a much lower level than on the trolley, and if the gas got so heavy as to reach down that far the mines would not be safe to work under any circumstances. These particular locomotives are operating under probably the most adverse conditions of any locomotives in the country to-day, and among other things they are taking the empty cars up a 14 per cent. grade.

Our first experience was that one of the locomotives was smashed almost completely by running away going down loaded. That was the fault of the mechanical construction, of course, and was due to the fact that owing to the weight of the battery the mechanical part was too light. In other words, in the case of a 7-ton locomotive there were 4 tons of battery and only 3 tons of locomotive, and that required certain forms of construction, and we got it too light.

Since being repaired, these locomotives have been operating practically every day, handling from 80 to 95 wagons per day, gathering them from the rooms, collecting them, and delivering them to the trolley locomotive on the cross heading.

We have not as yet kept any figures of cost, because the repairs have been trifling; in fact, there have been no repairs to the battery except one set of lead connections which were burnt out by a short-circuit. The cost of these locomotives was almost exactly twice that of trolley locomotives for the same capacity and same work, and the work they will do depends almost entirely on the facilities for charging. These locomotives are charged at night, 7 hr. each, and we take an hour's boosting at noon. Without that hour's boosting it would reduce the capacity about 20 per cent. We are now arranging to charge the locomotive right at the work, at the point where it delivers the load to the trolley locomotive, which will enable us to charge from the trolley through resistance during the 5, 10, or 15 min. periods the locomotive is waiting for the cars. We

expect to get ultimately to the capacity of the trolley locomotive, which we have not been able to do yet.

I do not believe that the storage-battery locomotive will be an economical proposition compared to the trolley locomotive in coal mines under ordinary conditions. I think it will be very useful for advance work, and where there is a small amount of work on a long heading, or possibly in opening the mine. I believe that where it is required on account of safety, it will require a considerable amount of development before it will pass the requirements of the Bureau of Mines as a safe electrical proposition. We consider their requirements for mining machines to be very severe; not only must the motors be flame proof, but all the controlling apparatus, fuses, and things of that sort must be either operated under oil or very completely inclosed.

I may say that the cost of these locomotives was \$4,000 each, of which the battery was exactly half, and the battery we used is the cheaper of the two batteries in common use. The other one would cost \$1,000 more for the same capacity.

Testing and Application of Hammer Drills

Discussion of the paper of BENJAMIN F. TILLSON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 505 to 528.

T. E. STURTEVANT,* New York, N. Y.—Mr. Tillson's paper on hammer drills is of much interest to me. Early in 1909 I was invited to make tests of our drills at the Franklin mine of the New Jersey Zinc Co.

The first drill that we tested gave very poor results. This was a drill having a hammer piston $1\frac{3}{4}$ in. in diameter and $3\frac{1}{4}$ -in. stroke, the hammer piston weighing about 6 lb., the $1\frac{3}{4}$ in. diameter being exposed to pressure at either end. Although this drill has given excellent results as a sinking drill, its performance as a stoping drill was poor. This was one of the surprises of the limestone as spoken of by Mr. Tillson.

This led me to further investigation, and a second drill was developed, with a much lighter piston and a somewhat longer stroke, that gave better results. In the design of this drill, no attention was paid to the cost of construction, our whole attention being given to the increase in drilling speed and reduction of the quantity of air used. The first drill of this type had a piston weighing about $2\frac{1}{2}$ lb. and a stroke of $3\frac{1}{4}$ in. The piston was made with a hammer-bar extension, and the cylinder was made with two diameters of the bore, one in which the hammer-bar extension moved and the larger one in which the piston head moved. The air was taken in at either end of the larger bore, thus giving a greater area to the

* Chief Engineer, McKiernan-Terry Drill Co.

piston on the forward stroke than on the rearward stroke. The pressure area of the piston for the forward stroke was the 2 in. diameter. The pressure area for the inward stroke was 2 in. diameter less the $1\frac{3}{8}$ in. diameter of the hammer bar. The performance of this drill was much better than our first drill, the air consumption being less and the drilling speed slightly greater. We then increased the length of the stroke to $3\frac{1}{2}$ in., this giving better results. Again we increased the stroke to $5\frac{1}{2}$ in. without any gain over the drill with the $3\frac{1}{2}$ -in. stroke. Through a number of tests we gradually reduced the length of the stroke until we found that the highest drilling speed was attained with a stroke of $3\frac{7}{8}$ in.

While I note the factor of efficiency as given by Mr. Tillson has been exceeded by other makes of drills, the work of our drills in the mine was satisfactory. The usual day's run was from 14 to 18 6-ft. holes per shift, and in two separate instances there were 26 6-ft. holes drilled and the heading fired in one shift.

The question of the loss of efficiency with an anvil block, as well as that of the position of the steel in relation to the rock during the moment of impact, is fully covered by Mr. Tillson. In our drills, I can note no difference in the drilling speed with or without the anvil block. We find that with the steel loose enough so that it will ring at each impact of the hammer, the highest drilling speed is attained.

The trial of drills by the mine management of the New Jersey Zinc Co. has always been conducted with the utmost fairness, and at no time could these trials of drills be considered a contest, as strict privacy was given to each manufacturer, and while careful record was kept of the performance, these data were not available to any, except the owners of the drills and the mine management. I have been a frequent visitor at the mines since 1909, and I have at no time there seen the construction of a competing drill or from them had any information as to its performance. This, I believe, is their general rule to all who have drills at their mine for trial.

The drill manufacturers can furnish hammer drills with a high cutting speed, but the users have the human element to contend with and I believe that the best results can be attained with a bonus system.

H. M. CRANKSHAW, Lansford, Pa.—Mr. Tillson stated that the efficiency of these drills has been improved about four times. Does all of this improvement come from the drills, or has any of the improvement come from the air? It has been my experience that the capacity of a drill can very often be doubled by increasing the capacity of the compressors and by putting in good air lines. They have the same plant now that they had in 1907, but the question arises, are the air lines any better now than they were in 1907? These figures, I take it, are figures over a series of time, and not figures taken from a test.

There is another rather curious point here, about the bit with the broken wing. I have seen it stated that a common X-bit with the two edges which were in line raised $\frac{1}{8}$ in. higher than the other edge, would give better results in drilling than the ordinary X-bit. Did Mr. Tillson follow up this seeming advantage he got with the broken wing?

BENJAMIN F. TILLSON.—The question of the increase in drilling efficiency is covered by this summary of the tests. You can follow it in the tests. The air pressure at the drill during the test is tabulated, and approximately the same air was used in the tests, so that the question of drill-plant equipment has not in any way affected it. As a matter of fact, all of these tests have been conducted with the same installation of air line in our shafts and the same pressure in the mine.

In regard to the broken bit, I probably missed making the point I intended to. It was that even though the bit was broken or shattered, or soft or plastic, yet some particular drill steels might drill faster than other drill steels with perfect bits and under the same standard testing conditions of machine, air pressure, ground, and drill gauge. Although many factors enter into the problem it seems that the synchronizing of the vibrations in the drill steel may be such as to increase the drilling speed, even though the bit were broken or shattered, to a figure in excess of the customary speeds.

R. M. CATLIN, Franklin Furnace, N. J.—At first sight, these tests of Mr. Tillson's seem to indicate a path along which some valuable results might be arrived at, but as these tests progressed further, the number of variables and unknown quantities in the equation became so great that it rather leaves the question somewhat as it was in the beginning.

At first, it seemed clear that if the drill point was pressing against the rock at the instant the blow was delivered, or, more correctly, at the instant that the impulse liberated by the blow reached that point, the greatest cutting effect would occur; that is to say, that in a drill it takes an appreciable time for the impulse to travel through the length of the metal. Now, if we have the hammer striking the drill when that impulse is rising, and the drill is pressing against the rock, we get the maximum cutting effect, whereas if the time is so arranged that the blow of the hammer meets the impulse coming back, you get the work done at the other end of the drill. In actual practice we have seen that as you put your hand on the drill you find a point which is intensely hot, and a few inches farther down quite a moderate temperature, and still farther down another center of temperature, which evidences the points at which, I think you might say, these two impulses overlap each other, nodes, so to speak. It seems that if one could test the steel as to its capacity for transmitting vibrations, and from a pile of steel select certain drills that had the same capacity, and then regulate his machine so as to deliver the blow at the

right time, that with a very poor drill you could get a very remarkable result.

That seems to be so. The next thing is to work it out. I know of no way of determining the capacity of steel for transmitting impulse except to actually test each individual piece, and then, after you have run a little while, that capacity changes. If you are running it at a velocity of 1,800 blows a minute, a few blows more or less will change the rythm of the whole thing, so that the point is more of academic interest than of special value. I merely mention it to account for some of the vagaries which Mr. Tillson mentioned. It obviously cannot be better to drill with a dull drill than a sharp one, but the cause is more obscure than it seems, I think.

L. C. BAYLES,* Phillipsburg, N. J. (communication to the Secretary†).—This paper brought up several points of interest to any one connected with mining, but the most important is the determination of the relative desirability of a drill, as practiced by the New Jersey Zinc Co. for the past six years.

On pp. 511 and 512, Mr. Tillson develops what appears to be a perfectly fair formula for making such comparisons. But as it is impossible to determine the upkeep by a short test and since both this and the first cost give such a small figure when divided over the total footage of these machines, Mr. Tillson is no doubt justified in omitting "M" and reducing the formula to the simple expression "Factor of desirability"

$$F = \frac{1}{D+P}$$

With this simple and fairly accurate formula available, it is hard to understand why the New Jersey Zinc Co. should have made up and adopted the empirical formula $F = \frac{1}{DP}$ which is no simpler to use and is obviously wrong, as it gives too much importance to the relatively small item of the value of the compressed air used.

On p. 514, Mr. Tillson takes two hypothetical cases and shows that their relative comparison remains about the same when compared by one or the other formula. But unfortunately he assumed two cases which both showed the same mechanical efficiency, that is to say, drilling an equal amount per unit of power input. Under such conditions naturally the fastest drill would show the highest "Factor of desirability" no matter which formula was used.

If Mr. Tillson had assumed that the slow drill would do 8 in. per minute, but that the other figures all remained the same, he would have seen the difference in the answers obtained by the two formulas.

* Ingersoll-Rand Co.

† Received Mar. 23, 1915.

Using $F = \frac{1}{D+P}$ we would find the fast drill was 17 per cent. more desirable. But using the empirical formula it would look as though the slow drill was 7 per cent. the best.

The above assumptions are entirely within the limits of possibilities. If we select two drills from the table on p. 510, say the ones marked "M" and "XO," we get entirely different results according to the method of calculation. Using $F = \frac{1}{D+P}$ we get "XO" or the fast machine about 4 per cent. more desirable. But by the empirical formula it appears as if the slow machine marked "M" was close to 20 per cent. the best.

Some Defects of the United States Mining Law

Discussion of the paper of COURTENAY DEKALB, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 331 to 337.

HORACE V. WINCHELL, Minneapolis, Minn.—Mr. DeKalb summarizes the reasons which he believes are important for the revision of the mining laws, as follows:

"1. The conceptions as to the characteristics of orebodies that were held at the time the statute of 1872 was enacted have since proved to be in many respects erroneous."

We all appreciate the force of that suggestion. The idea of the old prospector who made rules and regulations in the early days, which were attempted to be incorporated, so far as possible, in the Mining Act of 1872, was that all of the veins in a certain district outcropped upon the surface, or approximately parallel to each other upon the surface, and maintained that parallelism beneath the surface. Therefore, it seemed perfectly proper and natural for each man to stake his claim upon an outcropping, and have the right to follow that vein in working downward, with the general expectation that his workings would always remain just as far away from those of his neighbor as the outcroppings upon the surface. This does not happen to be the case. Dr. Raymond has very ingeniously referred to the subject in one case, in words which read as follows:

"If all mining properties presented this beautiful simplicity of structure (ideal location of ideal vein), and all mining locators exhibited a corresponding simplicity of purpose, the application of the law would be easy, but the naïveté of the statute fares badly between the freaks of nature and the tricks of man."

Mr. DeKalb further says:

"2. The operation of the existing law has been found to work unnecessary hardship upon many claimants to mineral land."

The particular feature of the law which I think was in the mind of Mr. DeKalb in writing that paragraph was the requirement of a discovery before location. No other country, so far as I am aware, requires the prospector to make an actual discovery of mineral in place before staking out his claim. The more you think of it, the more absurd it becomes. A prospector is a man searching for mineral, for a vein, for a deposit. He wishes to find something to which he can later obtain title. He must necessarily be protected in his possession while he is searching for his vein, but under the present law he is a trespasser upon the public domain until he has found his vein. Twenty-five years ago it was perhaps an easy matter to make a discovery without any prospecting work. To-day it is exceedingly difficult. Twenty-five years ago there were no National Forests, there were no geological or mining experts appointed by the government roaming over the country and closely scrutinizing the prospective work of the attempted locator. To-day if a man makes a location within the limits of the National Forest, within a few days a geological expert comes along and says to him, "What kind of a location have you? Let me see. If it is a lode claim, have you a vein in place? If it is a placer claim, have you rich enough material to pay for working?" If you have neither of these, then you have no location, "skidoo." A prospector must be assured of a reasonable possession of title while he is searching for his vein, as well as subsequently to its discovery. Under our present law he has no such right.

The third point made by Mr. DeKalb is as follows:

"3. The existing statute fails in large degree to encourage development of mineral resources."

This is linked up with the preceding point. So far as the law discourages the prospector by denying him the right to stake a claim wherever he desires to work, and there within the limits of his claim search for the vein or deposit, the prospector will not be so numerous as in the past, and will not do the work of pioneering which has been so valuable heretofore.

The fourth point made by Mr. DeKalb is:

"4. The existing law often produces conflict between placer and lode-claim rights, working unnecessary injustice to placer claimants."

This is particularly true, because of the defect in the law which makes it possible for a placer claimant, years after acquiring title, to be compelled to defend himself against innumerable contests and court cases on the part of lode claimants who come into the district after railroads have been built or reduction plants established, mines developed, and who finds a vein crossing patented placer grants. A lode claimant may at any time institute a contest, declaring that the applicant for the placer claim

had not at the time he made application for patent declared the existence of veins within his ground, and therefore was guilty of fraud, and had not paid the price per acre which the government exacts for lode claims. If the later applicant is successful, the result is under our law as at present constituted that he takes away from the placer owner property to which he had for 20 years supposed himself entitled, and becomes by reason of the success of his contest the owner of the veins which lie within the placer ground.

The fifth point brought forward by Mr. DeKalb is:

"5. The methods of establishing the rights of locators on United States mineral lands are defective, and are not uniform throughout the States within which such lands exist."

This point is discussed at some length in the paper, and it is true that the different States have somewhat different provisions regarding the rights of locators and the particular work required in perfecting a location.

The Capillary Concentration of Gas and Oil

This discussion of the paper of Chester W. Washburne should have been printed following the remarks of H. A. Wheeler, in *Bulletin* No. 100, April, 1915, pp. 835 and 836, but was inadvertently omitted.

ROSWELL H. JOHNSON:—There is one reason why the hydrostatic interpretation of gas pressure is not wholly acceptable. As we go down in the sands we seem to get excesses of gas. Particularly in this field it becomes excessive in quantity relative to the amount of water in the sand. If this holds in other fields as well it must mean that there has been an ascension of water, because these sedimentaries when laid down, were filled with connate water. If we do get an ascension of connate water, produced by some such phenomena as its expansion by heat or by the formation of new gas by pressure upon organic materials at great depth, we can hardly expect the pressure to be controlled merely hydrostatically.

* In reply to Mr. Wheeler's inquiry as to gas wells with pressure higher than hydrostatic pressure, I would cite the original wells given by Orton on which he based the hydrostatic theory. He used in his calculation a head of 600 ft. above tide, assuming the outcrop of the Manitoulin Islands to be in Lake Superior, whereas the Trenton does not outcrop at Lake Superior. The Manitoulin Islands are in Lake Huron, the level of which is 580 ft. above tide.

No allowance is made for the lower density of the fresher water near the outcrop, the weight taken for the whole column being the water of high density found at the wells.

While Dr. Orton admits that "defects of porosity would abate the pres-

* Communication to the Secretary

sure" he assumes no such "defects" from the gas wells to the outcrop, a violent assumption, since Phinney states that the Trenton is "close or only feebly porous" over large areas and Bownocker refers to a well in Wood County that had no gas until a depth of 50 ft. into the Trenton, although most of the production was obtained above that horizon. He also states that "the pay commonly ranges in thickness from 5 to 30 ft., it is more porous and commonly has a darker color than the inclosing rocks and hence is easily recognized."

Another objection to Dr. Orton's calculation in this case is, that in order to get the proper pressures to appear for Indiana, Phinney takes his head from the Trenton at the Cincinnati anticline rather than Lake Superior. If for the Trenton in the Indiana field, why not for the Trenton in the Ohio field?

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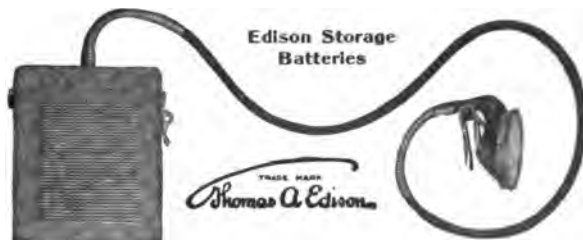
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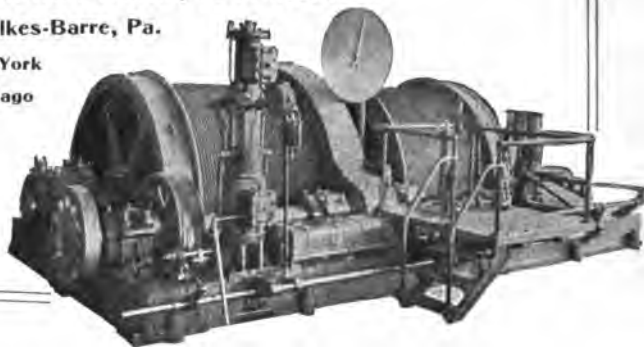
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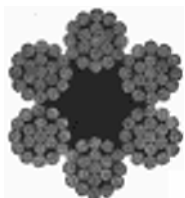
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.....	47 W. L. Austin.....	Leaching Copper Products at the Staptoe Works....	.10
.....	46 S. E. Bretherton.....	The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process.....	.10
.....	60 Stuart Crossdale.....	Leaching Experiments on the Ajo Ores.....	.10
.....	944 Chase Palmer and Edson S. Bastin.....	The Role of Certain Metallic Minerals in Precipitating Silver and Gold.....	.10
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.....	965 Frederick Laist.....	Roasting and Leaching Tailings at Anaconda, Mont.....	.20
.....	975 J. C. Fobles.....	The Precipitation of Copper from the Mine Waters of the Butte District.....	.10
.....	28 Frederick Laist and F. F. Friok.....	Precipitation of Copper from Solution at Anaconda.....	.10
.....	27 William F. Ward.....	"Playa" Panning on the Cauca River.....	.10
.....	44 A. W. Allen.....	The Descriptive Technology of Gold and Silver Metallurgy.....	.10
.....	33 E. Horton Jones.....	Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd. (Bound in Cloth)....	2.00
.....	63 W. W. Norton.....	A Comparison of the Huntington-Heberlein and Dwight-Lloyd Processes.....	.10
.....	981 Frank H. Corwin and Selden S. Rodgers.....	Increasing the Efficiency of MacDougall Roasters at the Great Falls Smelter of the Anaconda Copper Mining Co.....	.20
.....	943 G. S. Brooks.....	Notes on the Formation of Ferrites in Roasting Blends.....	.10
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.....	56 C. D. Demond.....	Economy and Efficiency in Reverberatory Smelting.....	.10
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.....	1000 Dorsey A. Lyon and R. M. Keeney.....	The Smelting of Copper Ores in the Electric Furnace.....	.10
.....	964 Woolsey McA. Johnson.....	The Reducibility of Metallic Oxides as Affected by Heat Treatment.....	.10
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.....	996 C. W. Goodale.....	The Great Falls Flue System and Chimney.....	.40
.....	998 Edgar M. Dunn.....	Determination of Gases in Smelter Flues; and Notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont.....	.20
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AMERICAN INSTITUTE OF MINING ENGINEERS

29 West 39th St., New York, N. Y.

Smelter Construction Costs

*Unit Construction Costs from the New Smelter
of the Arizona Copper Co., Ltd.*

By E. HORTON JONES

152 pages, 6x9, Illustrated, \$2.00 (8/4) net, postpaid

Reprinted by special arrangement with the American Institute of Mining Engineers

SMELTER construction costs in the fullest detail are given in this book. The data are based on construction costs derived from the building of the Arizona Copper Company's new smelters at Clifton, Arizona, completed February, 1914.

In Chapter I—UNIT COSTS—are to be found the most elementary total unit costs which the accounts provide for; also the percentages to be added to an estimate for Engineering and General Expense.

In Chapter II—COMPARATIVE COSTS—these elementary costs have been classified, averaged and reported as labor and material unit costs. In such form, the labor unit costs, when properly applied to similar conditions as these, under which they were derived, are usable anywhere.

In Chapter III—COMPOSITE COSTS are given. They are unit costs built from several elementary units, and likewise units of larger dimensions and similar application, valuable for checking estimates and obtaining quick approximations of total costs.

In Chapters IV, V and VI are given the WAGE SCALE, MATERIAL PRICES, and a description of the conditions surrounding the making of every elementary unit cost, which will enable an estimator to judge of their use in any circumstances.

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JUNE, 1915



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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Occupation _____

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of the American Institute of Mining Engineers.

} Signatures of three
Members or
Associates.

Place of birth _____ Year of birth _____

Education, general and technical, when, where and how acquired,
with degrees, if any.

Dates	

[illegible]

_____ Signature

Dated _____ *191*

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

Sec. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

Harvard College Library

Aug. 14, 1916.

Bequest of

Erasmus Darwin Leavitt.

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PUBLISHED MONTHLY

No. 102

JUNE

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTSKY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STROUGHTON, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

SAN FRANCISCO MEETING

The 111th meeting of the Institute for the presentation and discussion of technical papers will be held in San Francisco, Cal., Sept. 16 to 18, 1915. A special train for members of this Institute and for the members of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the Society of Naval Architects and Marine Engineers, and the International Engineering Congress, will be run from New York City to San Francisco, leaving New York on Sept. 9.

All papers to be presented at this meeting of the Institute must be in the hands of the Secretary on or before June 1, 1915.

NOMINATIONS FOR OFFICERS

The co-operation of the members of the Institute is earnestly sought by the Committee on Nominations, recently appointed by the Board of Directors, in its work of formulating a ticket for officers and places falling vacant in the Board of Directors in February, 1916. The Committee must present the official ticket of nominations to the Board of Directors at its October meeting. Members are, therefore, urged to send promptly their nominations for officers and Directors in order that the Committee may have an opportunity to give them full consideration.

The officers to be elected at the annual meeting in February, 1916, are:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N. B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, William L. Saunders (not eligible for re-election).

Past President, Charles F. Rand.

Vice-Presidents, Thomas H. Leggett, District 0; Fred W. Denton, District 4.

Directors, John W. Finch, District 7; John H. Janeway, District 0; Edward P. Mathewson, District 5; Joseph W. Richards, District 2; George D. Barron, District 0.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, Crocker Bldg., San Francisco, Cal.

F. W. BRADLEY,

Chairman, Committee on Nominations.

BOARD OF DIRECTORS

Meeting of Apr. 23, 1915.—The Iron and Steel Committee was authorized in its discretion to defer the date of acceptance of papers for the Hadfield Research Prize to Nov. 1, 1916.

The sum of \$250 was appropriated as a guarantee fund to establish a Library Research Bureau in co-operation with the other Founder Societies.

The sum of \$50 was appropriated to the Puget Sound Section.

The sum of \$250 was appropriated to the San Francisco Section.

The sum of \$125 was appropriated to the St. Louis Section.

The matter of memorializing the New York State Constitutional Convention was referred to the New York Section of the Institute with authority to act as a Section only.

Upon recommendation of the Committee on Membership, 50 Members, 1 Associate, and 6 Junior Members were elected.

Upon recommendation of the Executive Committee of the Committee on Papers and Publications, it was voted that a footnote in connection with each of the papers published by the Institute shall give the official title and address of the author, and in addition thereto his collegiate degrees, if he so desires.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Apr. 10 to May 10, 1915:

William W. Elmer, Tuttletown, Cal.

Lawrence Addicks, Douglas, Ariz.

W. F. Durand, Stanford Univ., Cal.

S. P. Cooper, New York, N. Y.

R. R. Freeman, Jr., Wollaston, Mass.

Francis R. Pyne, Elizabeth, N. J.

George B. Pickett, Ophir, Colo.

R. V. Norris has been elected President of the Columbia School of Mines Alumni Association.

Robert C. Sticht has been elected President of the Australian Institute of Mining Engineers.

James T. Dixon is to make an inspection of the Crown mines on the Rand.

J. F. Mitchell-Roberts has resigned his position as Chief Engineer for the Wilfley Machinery Co., of London, to take up private work.

Durward Copeland has resigned the directorship of the Rolla School of Mines, and is preparing for a trip to England, the Straits, and other tin-producing countries to study the tin situation throughout the world for South American interests.

Chester W. Washburne has resigned from the U. S. Geological Survey to engage in private work, specializing in the examination of oil, gas, and coal properties. Returning from South American fields he has gone to Africa.

Harold A. Morrison, formerly of Dedrick, Cal., has accepted a position in the mill of the Aurora Consolidated Mines Co., Aurora, Nev.

Henry S. Munroe will retire from the professorship of mining in the Columbia School of Mines on June 30, after 38 years of service.

Allen H. Woodward was chosen Chairman of the Board of Directors of the Woodward Iron Co. at its meeting on Apr. 22 in New York, at which time the position was created. Mr. Woodward was advanced from Vice-President and General Manager.

Russell T. Cornell has removed his offices to 80 Maiden Lane, New York, adjoining those of Ricketts & Co., Inc., by whom he has been retained as associate mining and consulting engineer. His new association will not in any way affect his present status as an independent engineer.

Fletcher G. Downs, who has been with the Instituto Geologia y Perforaciones in Uruguay for the last two years, has returned to the United States and his present address is Chatham, N. J.

Benjamin Magnus, who has been General Manager of the Mount Morgan mine, Queensland, for several years, has resigned that position, and also his position as consulting engineer to the Electrolytic Refining Co., Port Kembla. He intends to leave Australia about June 15 for the United States.

Solomon Le Fevre, who has been with Witherbee, Sherman & Co. for the past nine years as Assistant General Manager, General Manager, and general mining engineer, announces that he is available as consulting mining engineer, with permanent address at Forest Glen, Ulster County, N. Y.

Clarence Boyle, Jr., formerly District Sales Manager of the Taylor-Wharton Iron & Steel Co., Scranton, Pa., has become associated with Clarence Boyle, Inc., wholesale lumber, Chicago.

John Howe Hall, 2 Rector Street, New York, has discontinued his practice as consulting engineer, and has entered the employment of the Taylor-Wharton Iron & Steel Co., High Bridge, N. J., as metallurgical engineer.

J. V. N. Dorr announces that the New York office of the Dorr Cyanide Machinery Co. will be at 17 Battery Place after May 1, 1915.

Lawrence Addicks was elected President of the American Electrochemical Society at Atlantic City on Apr. 22, 1915.

Lloyd B. Smith, of the Associated Geological Engineers, has returned to Pittsburgh after spending three months in the oil fields of Mexico and Central America.

Frederic Keffer, for the past 17 years associated with the British Columbia Copper Co., Ltd., of New York and British Columbia, as General Manager, consulting engineer and geologist, has formed a partnership with Henry Johns, and opened an office at 214 Hutton Bldg., Spokane, Wash., for consulting mining work.

Richard Tarantous has accepted the position of Assistant Superintendent of the Chisos Mining Co., operating a cinnabar property at Terlingua, Texas.

Maurice S. Brandt has accepted a position as Chief Engineer and General Manager for the Seneca Consolidated Gold Mines and the Van Winkle Group, at Seneca, Plumas County, Cal. He will retain his position as consulting engineer for the White Eagle Mining & Smelting Co.

Carl N. Anderson has accepted the position of engineer for the Alaska Consolidated Copper Co., Strelna, Alaska.

William C. Schmidt, Jr., has accepted a position with the New York & Honduras Rosario Mining Co., San Juancito, Honduras.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Opening for man as head of laboratory of large brass manufacturing plant in Ohio; must also be competent to manage foundry making brass castings, following practice of malleable iron castings and insulating materials. Good opportunity for right man. No. 36.

Professorship in Mining at a Western university; requires man competent to give instruction in ore dressing, in metal and coal mining, and in zinc smelting, and capable of gaining confidence of mining interests. Salary \$2,200 to \$3,000. No. 37.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, graduate engineer, aged 44, with over 20 years' experience in connection with large coal and coke operations in different parts of the country, desires position as general superintendent of large coal company, or will take an interest with the management in a smaller company. No. 206.

Sales manager, technical, mining and mechanical, broad experience advertising and selling, will be open for engagement in a few months. Nation-wide acquaintance among buying powers at mines, mills, and smelters. No. 217.

Member, American, graduate Lehigh University, with 14 years' experience, largely in bituminous coal mining; has had active charge of difficult mining problems such as gob fires in gaseous mines, explosions, etc., as well as many kinds of electrical machinery and mechanical mining plants. Well qualified by training and actual experience to undertake management of concern requiring broad engineering experience and ability to organize and oversee productive work. New engaged as general manager of coal-mining company abroad. Is planning trip to Panama Fair in June or July and can arrange personal interview at any point in the U. S. *en route* New York to San Francisco. No. 218.

Member, available during summer as first assistant to consulting engineer engaged on any kind of examination work. Technical graduate in mining geology and metallurgy. Several years' experience. Speaks Spanish. High-class references for any kind of work. No. 219.

Member, technical graduate, aged 36, 15 years' experience in field, underground, stamping mill, concentrating, cyaniding, gold metallurgy, and mine examination work—from the bottom to general superintendent and manager. Preference: for residential work in latter capacity anywhere north of Mexico. Best references. Would go to New York or Boston for personal interview. No. 220.

Member, technical graduate, aged 41, with 17 years' Western and South American experience. Has held position as superintendent of prominent Western copper mine, also of large copper property in South America. Wants position operating. Specialty, underground methods for large tonnage and low costs. References. No. 221.

Member, technical graduate, aged 28, married, thorough experience in all-sliming cyaniding, is open for engagement as metallurgist, or other responsible position. Interview at San Francisco Meeting, A. I. M. E., in September. No. 222.

The undersigned member is open for engagement; is a graduate of School of Mines, Columbia University; has had 23 years' experience in mining and mechanical lines: as mine surveyor, engineer, and superintendent; as designing, constructing and manufacturing engineer; as instructor and college professor. Edward B. Durham, 2227 Ward St., Berkeley, Cal.

Member, graduate geologist, single, speaks Spanish. Five years' experience in coal and petroleum geology. Familiar with the oil regions of the Mid-Continent fields and Mexico. References. Available July 1. No. 224.

Member, specializing in prospecting and development work, with 10 years' experience in the West and Canada, desires position. No. 225.

Member, finishing an engagement during May on mine development and construction work at mining properties in Virginia, will be open for other engagement after June 1. Extensive experience in mining and civil engineering with large mining companies and in examination of mining properties. Best of references. No. 226.

Member, with varied experience in developing and operating coal, iron, and lead mines; 18 years in present position, as mining engineer; for climatic reasons desires position in Northern or Eastern States, or Canada. Good references. No. 227.

IRON AND STEEL COMMITTEE

JOSEPH W. RICHARDS, *Chairman*.
 J. E. JOHNSON, JR., *Vice-Chairman*.
 ARTHUR S. CALLEN, *Secretary*, 453 Chestnut St., So. Bethlehem, Pa.

SUB-COMMITTEES

IRON ORE

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CHEMISTRY, PHYSICS, AND METALLOGRAPHY

HERBERT M. BOYLSTON, ALBERT SAUVEUR, *Chairman*.
 HENRY M. HOWE, E. GYBBON SPILSBURY, WILLIAM R. WEBSTER.
 LEONARD WALDO,

An interesting and profitable meeting of the Iron and Steel Committee was held at the Engineers' Club, on Friday, Apr. 16, at which 12 members were present.

The Committee recommended to the Board of Directors deferring the date of acceptance of papers for the Hadfield Research Prize to Nov. 1, 1916.

The Committee petitioned the Bureau of Standards to direct some of its attention to determining lacking but badly needed data concerning iron, such as its specific heat in the liquid state, latent heat of fusion, electric conductivity when molten, heat conductivity up to the melting point, for pure iron and some typical cast irons and steels.

Reports from Chairmen of sub-committees showed three papers *in hand* and seven papers *promised* for the next meeting of the Institute, also good *prospects* for seven more in the near future, and *possibilities* of many more for the February meeting of the Institute.

A poem by Professor Sauveur, printed on the menu card, is given herewith.

JOSEPH W. RICHARDS,
Chairman.

SATIRE ALLOTROPIQUE

ALBERT SAUVEUR

Fer bête, conception du grand allotropiste,
 Contre toi, profitant de la sortie du maître,
 Se rangent tes rivaux faciles à reconnaître,
 Mirmidons entraînés par propos carbonistes.

Hadfield l'avait bien dit: le carbon seul durci.
Alpha, bêta, gamma, composants inutiles,
Ne pouvant arrêter le moindre projectile,
Doivent être relégués, et cela sans merci.

Arnold, chacun le sait, toujours a préféré,
Par lui seul découvert, un carbure éphémère,
Une formule étrange, véritable chimère,
Création fabuleuse qui ne peut qu'égarer.

Quant à Henry M. Howe, de nous tous le mentor,
Avec une justice qu'on rarement rencontre,
Il pèse sans faveurs les pous et les contres,
Et nous dira bientôt qui a raison ou tort.

Stead est décourageant par sa neutralité.
Tschernoff et Belaiew, fidèles amis slaves,
Des nouveautés anglaises ne seront pas esclaves,
Ce qui prouve après tout pour leur mentalité.

Guillet, Grenet, Charpy, Henri Le Chatelier
(Pour Frémont ces questions ont trop de théorique,
Son fort étant plutôt celui de la pratique)
Ne sont pas de bêta des ardents chevaliers.

Rosenhain avec lui semble jouer cache-cache:
Il l'attrape, il le tient, proclame qu'il est dur.
On accourt, on regarde, cette fois on est sûr,
Mais hélas aussitôt, voila qu'il le relâche.

C'est un slip band dit l'un, un simple glissement.
Du ciment dit un autre, un rien allotropique,
Pas même un métastable, un point mathématique,
Un décristallisé à chasser vite ment.

Benedicks dans gamma le veut ensevelir,
McCance l'écrourir, Edwards en faire des mâcles:
Le malheureux bêta devant cette débâcle,
Se soumet à son sort, se résigne à mourir.

Il attend son destin, triste et mélancolique,
Abandonné de tous, même de martensite,
Qui prétend n'être plus que rides d'austénite.
Il deviendra amorphe. Adieu son point critique.

Pour comble de malheur, et sans le consulter,
Un docteur Suédois le veut mettre sous terre,
Assisté dans sa tâche, par Monsieur Carpenter,
Car, dit-il, il est mort, on le peut constater,

Etant de phlogistique tellement dépourvu,
Qu'on ne sait en tirer le moindre point thermique,
Ni dans sa courbe voir la plus petite crique.
Tel manque de chaleur s'est très rarement vu.

Le verdict est rendu, le sort en est jeté:
D'un certain diagramme if faudra qu'on efface,
(Car on doit obéir à la règle des phases)
Ce déséquilibré, enfantement raté.

Quand vous aurez, Messieurs, fini vos beaux discours,
Et que l'heure viendra, pour gagner votre pain,
De durcir vos aciers et d'assurer vos gains,
Il faudra que bêta vienne à votre secours.

ST. LOUIS LOCAL SECTION

*Executive Committee*ARTHUR THACHER, *Chairman*R. A. BULL, *Vice-Chairman*WALTER E. McCOURT, *Secretary-Treasurer*, Washington University,
St. Louis, Mo.

H. A. BUEHLER

R. R. S. PARSONS

HERBERT A. WHEELER

On the afternoon of May 1, 1915, 42 members of the St. Louis Section, as the guests of E. S. Gatch, President, and his associates in the Granby Mining & Smelting Co., visited the new zinc smelter of that company at Roselake, Ill.

The annual meeting, which was preceded by a dinner at the St. Louis Club, was attended by 54 members and guests. The Chairman, H. A. Wheeler, presided. The Chairman rendered a report on the activities of the Section during the past year and the Treasurer presented a financial report. The following Executive Committee was elected: Arthur Thacher, Chairman; R. A. Bull, Vice-Chairman; W. E. McCourt, Secretary-Treasurer; H. A. Buehler, R. R. S. Parsons. H. A. Wheeler, as retiring Chairman, retains membership in the committee.

W. L. Saunders, President of the Institute, made an address. H. W. Gepp, of Australia, described the lead and zinc deposits of Broken Hill, Australia, and V. H. Hughes, of Tulsa, spoke on the geology of the Cushing oil pool, Oklahoma.

WALTER E. McCOURT, *Secretary*.

CHICAGO LOCAL SECTION

*Executive Committee*ROBERT W. HUNT, *Chairman*J. A. EDE, *Vice-Chairman*HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural
History, Chicago, Ill.

F. K. COPELAND

G. N. DAVIDSON

The Chicago Local Section of the Institute met Friday, Apr. 30, at the Chicago Engineers' Club. There were 29 members and guests present. In the absence of the Chairman and Vice-Chairman, F. K. Copeland presided.

After the dinner the President of the Institute, W. L. Saunders, spoke of several lines of recently inaugurated Institute activity and of the mining engineering profession in Chicago.

Carl Scholz spoke of business conditions which are seriously interfering with the prosperity of the coal-mining industry in Illinois and of attempts to remedy the undesirable conditions which are now under way. It was voted that the Executive Committee should take under consideration the desirability of the Section co-operating with other organizations in the matter.

John Bruner, of the Illinois Steel Co., showed and explained two reels of moving pictures of the steel mills at Gary.

The present officers of the Section were unanimously re-elected for the season 1915-16.

HENRY W. NICHOLS, *Secretary*.

AFFILIATED STUDENT SOCIETY NOTES

Officers as follows have been elected by Affiliated Student Societies of the Institute:

School of Mines Society, University of Minnesota.

President, Adolph Dovre.
Vice-President, Oscar Lee.
Secretary, Edwin Sweetman.

Mining Society, Lehigh University.

President, S. Martin.
Vice-President, H. E. White.
Secretary, K. A. Lambert.
Treasurer, R. L. Colby.
Curators, H. E. Sanford, A. S. Constine.

Student Branch of the A. I. M. E., Ohio State University.

President, Ellsworth H. Shriver.
Vice-President, Raymond T. Whitzel.
Secretary, William A. Curran.
Treasurer, William A. Heimberger.
Director-at-Large, Peyton Y. Dooley.

Mining Engineering Society, University of Arizona.

President, F. A. Luis.
Secretary, E. J. Renaud.

Associated Miners, University of Idaho.

President, W. N. Ellis.
Secretary, Clarence A. Sylvester.

The annual Short Session for mining men at the University of Washington closed its eighteenth year on Apr. 2, 1915. In many respects the session was the most successful in years, 38 students registering for the work, two of whom were women. The oldest student was 52 years old, the youngest 19. Eleven of the men had previously attended some college and three were graduates. Quartz mining subjects appealed to the greater number, with placer mining next in interest. Next year the nineteenth session will begin on Jan. 3 and run for three months.

Following the close of the Short Session the junior and senior classes of the College of Mines, together with a few of the short-course men, visited the Index and the Blewett mining districts of the State. Several days were spent at Index studying the Sunset, the Copper Bell, and the granite quarries of this section. At Blewett, which is the oldest mining camp in the State, gold mining and milling and the geology of the district were investigated.

In connection with the "open house" of the various engineering departments of the University on Apr. 15, the laboratories of the College of Mines were in full operation. Seniors and juniors ran the various divisions of work while freshmen and sophomores acted as guides. Among the special features of the performance were the operation of the stamp battery, crushing and concentrating machinery of the mill; rock drilling; mine timbering; first-aid and mine-rescue work; mineral and rock exhibits; fire and wet assay methods; cyanide and flotation processes; and metallurgical demonstration.

Between 250 and 300 students of the College of Engineering and the College of Mines visited the White River hydro-electric plant of the Puget Sound Traction, Light & Power Co. at Dieringer, Wash., with a rated capacity of 20,000 kw., and the Stuck River project near Auburn, Wash. The latter project involves the creation of a new drainage channel for the White River, which overflows its banks and floods an adjacent rich agricultural district. The work is being done by two counties at a cost of \$1,500,000.

SAFETY FIRST AT LIBERTY BELL MINE

The first annual safety first dinner of the Liberty Bell mine was held on Apr. 8, 1915, to celebrate "a year without a fatal accident and with fewer serious accidents."

The *menu*, which bore the inscription "Good food is a start toward health and safety," included not only the substantials but also plenty of the frills, and in reading it over one suspects that there was a near approach to a surfeit of good things.

Charles A. Chase and others from the mine, mill, and hospital gave short talks on safety and health.

Music also enlivened the occasion, the program including congregational singing of the following appropriate selection:

SAFETY FIRST

(Tune: "Tipperary")

When a man works underground, away up in the hills,
It does not swell his bank account to pay the doctor bills.
If he takes a little care while working in the mine,
His pocket-book is nice and fat and everything is fine.

Chorus:

It's the bad thing to get broke up; it's the wrong way to do.
It takes us off the payroll and hurts our feelings too.
So, it's goodbye Dr. Hadley, farewell Nurses too,
We are going to be more careful, and won't have to come to you.

"Safety First" is a watchword that runs the country thru;
Now if you'll take a tip from me, it applies to you.
This saying now you copy down, I'll tell you it's no lie.
And then just tell the doctor to kiss himself goodbye.

(Chorus)

You have heard so much about the European war,
It's Germany and England fighting out a bitter score,
Men out on the battlefield are being constantly shot down,
But you can die just like them by being careless underground.

(Chorus)

The Liberty Bell Gold Mine Creed is so good that we print it herewith in full.

LIBERTY BELL GOLD MINE CREED*What I will do for the New Year*

- I will think before I act.
 - I will pick down loose ground before starting other work.
 - I will sprag dangerous ground, to hold it until I can permanently timber it.
 - I will keep chute holes properly covered.
 - I will keep my ladder in repair and clean.
 - I will keep my station clean and safe.
 - I will keep my part of the track clean and not leave powder and caps exposed in the drifts.
 - I will remember that my life and my health are worth everything to me.
 - I will remember that my carelessness may endanger other men, besides me, and may make widows and orphans.
 - I will remember that I cannot be counted a good miner, if careless.
 - I will complain loudly if someone else endangers me.
 - I will talk freely to the boss about bad places and get his advice.
 - I will do all I can to make Liberty Bell the safest and best mine to work in.
-

FIELD MEET OF THE BUREAU OF MINES AND THE AMERICAN MINE SAFETY ASSOCIATION

Sept. 23 and 24 there will be held in Athletic Field, Panama-Pacific International Exposition, a demonstration and international contest of mine rescue and first aid, under the auspices of the Bureau of Mines, the American Mine Safety Association and the California Metal Producers, Association. Following is the preliminary program:

Sept. 23. 10:00 a.m. Mine-Rescue Demonstration. 2:00 p.m. First-Aid Demonstration. 4:00 p.m. Coal-Dust Explosion.

Sept. 24. 10:00 a.m. First-Aid Contest for Interstate supremacy. 2:00 p.m. Rescue Contest for Interstate supremacy. 4:00 p.m. Rock-Drilling Contest. 8:00 p.m. Award of prizes and souvenirs, Convention Hall.

EMMONS MEMORIAL FELLOWSHIP

Max Roesler has been selected as the first incumbent of the Emmons Memorial Fellowship by the Committee consisting of James F. Kemp, Chairman; J. D. Irving, Secretary; and Waldemar Lindgren. Mr. Roesler graduated from Sheffield Scientific School in 1907, and for five years after his graduation was employed for varying lengths of time on the Missouri Geological Survey, at the Barnes-King mine in Montana and on the geological staff of the Copper Queen mine, Bisbee, Ariz. He then was engaged in general examination work, and in the fall of 1913 took up graduate work in geology at Columbia University and completed the requirements for the M. A. degree. Since February, 1915, he has been instructor in geology at Yale. Mr. Roesler will take up for investigation some suitable problem in connection with geology, beginning July 1, 1915.

EMMONS VOLUME ON ORE-DEPOSITS*A continuation of the "Posepny" Volume*

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Contents

- Genesis of Certain Ore-Deposits. By S. F. EMMONS.
 Structural Relations of Ore-Deposits. By S. F. EMMONS.
 Geological Distribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by JOHN A. CHURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
 Torsional Theory of Joints. By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. BOYD, and GEORGE F. BECKER.
 Allotropism of Gold. By HENRY LOUIS.
 Superficial Alteration of Ore-Deposits. By R. A. F. PENROSE, JR.
 Some Mines of Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
 Genesis of Certain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS, G. F. BECKER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
 Influence of Country-Rock on Mineral Veins. By WALTER HARVEY WEED.
 Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
 Consideration of Igneous Rocks and Their Segregation of Differentiation as Related to the Occurrence of Ores. By J. E. SPURR. Discussion, by A. N. WINCHELL.
 Chemistry of Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
 Ore-Deposits near Igneous Contacts. By WALTER HARVEY WEED. Discussion, by W. L. AUSTIN.
 Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
 Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
 Geological Features of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. G. MILLER and W. L. AUSTIN.
 Osmosis as a Factor in Ore-Formation. By HALBERT POWERS GILLETTE.
 Ore-Deposits of Sudbury, Ontario. By CHARLES W. DICKSON.
 Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
 Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
 Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
 Genetic Relations of the Western Nevada Ores. By J. E. SPURR.
 Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
 Occurrence of Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
 Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. LEITE.
 Geological Relations of the Scandinavian Iron-Ores. By H. SjöGREN.
 Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J. BANCROFT.
 Distribution of the Elements in Igneous Rocks. By H. S. WASHINGTON.
 Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. EMMONS.
 Cognate Papers.
 Bibliography of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. C. FERGUSON.

The volume contains 1002 pages. Price, bound in cloth, \$5; in half-morocco, \$6. Both the Emmons and the Posepny Volumes on Ore-Deposits, bound in cloth, \$8; in half-morocco, \$10.

READY FOR DISTRIBUTION

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

COAL-MINE FATALITIES IN THE UNITED STATES, 1914. U. S. Bureau of Mines, Washington, 1915.

ABSTRACTS OF CURRENT DECISIONS ON MINES AND MINING, DEC., 1913, TO SEPT., 1914. Bull. 90, U. S. Bureau of Mines. Washington, 1915.

ART DE L'INGÉNIEUR ET METALLURGIE. (Extrait du volume III Tables Annuelles de Constantes et Données Numériques.) Chicago, 1914. (Gift of University of Chicago Press.)

DONNÉES NUMÉRIQUES DE L'ELECTRICITÉ, MAGNÉTISME ET ELECTROCHIMIE. (Extrait du volume III Tables Annuelles de Constantes et Données Numériques.) Chicago, 1914. (Gift of University of Chicago Press.)

[These two pamphlets are reprints from the "Annual Tables of Constants" of the portions relating (1) to Engineering and Metallurgy; (2) to Electricity, Magnetism and Electro-chemistry, thus placing at the disposal of the student at a small cost the essential standards.—W. P. C.]

SMELTING OF COPPER ORES IN THE ELECTRIC FURNACE. Bull. 81, U. S. Bureau of Mines. Washington, 1915.

HOUSES FOR MINING TOWNS. Bull. 87, U. S. Bureau of Mines. Washington, 1914.

MODERN PUMPING AND HYDRAULIC MACHINERY. By Edward Butler. London, 1913.

STAMP MILLING AND CYANIDING. By Francis Andrew Thomson. McGraw-Hill Book Co., Inc., New York, 1915. Price \$3. (Gift of Publishers.)

[NOTE.—This book, designed as a text, is an excellent presentation of the subject of the metallurgy of gold and silver developed from the author's class-room lectures. The book shows that the author has read widely and digested thoroughly the literature on the subject.

Part I is devoted to an exposition of the principles and practice of amalgamation, and the crushing and grinding of ores, with a comparison of the merits of the various crushing devices.

Part II is a general discussion of the subject of cyaniding, giving a brief history of the process, and descriptions of the successive operations and equipment used in carrying out the process.

Part III is devoted to the application of amalgamation, concentration, and cyanidation in different combinations to the treatment of gold and silver ores, this information being given by flow sheets of typical mills. This is a convenient and highly satisfactory method of presentation, since the information essential for the proper comparison of the practice in different mills and districts is thus given in compact and readily available form.

The comprehensive bibliography should prove a rich source of information and a valuable aid to any one desiring to make an exhaustive study of any branch of the subjects covered.—B. A. R.]

Geology, Mineralogy, Mineral Production

CALIFORNIA. MINES AND MINERAL RESOURCES OF IMPERIAL COUNTY, SAN DIEGO COUNTY. California State Mining Bureau, San Francisco, 1914.

MICHIGAN. MINERAL RESOURCES OF MICHIGAN, WITH STATISTICAL TABLES OF PRODUCTION AND VALUE OF MINERAL PRODUCTS FOR 1913 AND PRIOR YEARS. Publication No. 16, Geological Ser. 13, Michigan Geological and Biological Survey. Lansing, 1914.

ONTARIO. KIRKLAND LAKE AND SESEKINKA GOLD AREAS, SHOWING PROPERTIES OF OGILVIE AND MCKINNON TOWNSHIPS OF TECK, LEBEL, OTTO AND BOSTON, DISTRICT OF TIMISKAMING, ONTARIO. (Gift of Dane Mining Co.)

SWEDEN. SVERIGES GEOLOGISKA UNDERSÖKNING. Arsbok, 1912, 1913. Stockholm, 1914. (Gift of Sveriges Geologiska Undersökning.)

SWEDEN. BESKRIVNING TILL ÖVERSIKTSKARTA ÖVER SÖDRA SVERIGES LANDFORMER, AV STEN DE GEER. Stockholm, 1913. (Gift of Sveriges Geologiska Undersökning.)

VIRGINIA. MAP OF MINERAL TERRITORY TRIBUTARY TO NORFOLK AND WESTERN RAILWAY. 1909. (Gift of F. H. LaBaume.)

VIRGINIA. NEW RIVER CRIPPLE CREEK MINERAL REGION OF VIRGINIA. By A. S. McCreath and E. V. d'Inville. Harrisburg, 1887. (Gift of F. H. LaBaume.)

Non-Metallic Minerals

COAL FIELDS OF PIERCE COUNTY, WASHINGTON. Bull. 10, Washington Geological Survey. Olympia, 1914.

OCCURRENCE OF OIL AND GAS IN MICHIGAN. Publication 14, Geological Ser. 11, Michigan Geological and Biological Survey. Lansing, 1914.

PETROLEUM INDUSTRY OF CALIFORNIA. Bull. 69, California State Mining Bureau (with map). San Francisco, 1914.

PETROLEUM IN PAPUA, REPORT ON (WITH MAP). By Arthur Wade. 1914. (Gift of Author.)

PETROLEUM YEAR BOOK, 1914. London, 1914.

THE RARE EARTHS, THEIR OCCURRENCE, CHEMISTRY, AND TECHNOLOGY. By S. I. Levy. New York, 1915.

THE CALCITE MARBLE AND DOLOMITE OF EASTERN VERMONT. Bull. 589, U. S. Geological Survey. Washington, 1915.

Tunneling

SHORT AND LONG TUNNELS OF SMALL AND LARGE SECTION DRIVEN THROUGH HARD AND SOFT MATERIALS. By Eugene Lauchli. New York, McGraw-Hill Book Co., 1915. Price \$3. (Gift of the Author and the Publisher.)

[NOTE.—As the author states in the preface, this treatise on tunneling was prepared in the endeavor to supply practicing engineers and contractors with the information most usually needed in tunneling work, and to present to students the underlying principles indispensable to the solution of the many problems met with in driving tunnels serving miscellaneous purposes.

The importance of geological surveys in connection with tunnel driving is emphasized, and the effect on the driving methods of different geological formations is considered.

Descriptions are given of the different driving, timbering, lining, and haulage methods employed in the construction of tunnels of various cross-sections and lengths in hard and in soft ground, together with the costs of the different operations. The methods are illustrated by examples taken from actual practice.

Chapters on drilling equipment, compressor plants, ventilation, and rock temperatures are also included. Practicing engineers will undoubtedly find this work to be of assistance to them in the solution of their tunnel problems.—B. A. R.]

General

COAL TAR PRODUCTS AND THE POSSIBILITY OF INCREASING THEIR MANUFACTURE IN THE UNITED STATES. Technical Paper 89, U. S. Bureau of Mines. Washington, 1915.

PROBABLE EFFECT OF THE WAR IN EUROPE ON THE CERAMIC INDUSTRIES OF THE UNITED STATES. Technical Paper 99, U. S. Bureau of Mines. Washington, 1915.

Company Reports

BROKEN HILL PROPRIETARY CO., LIMITED. Report of 59th half yearly general meeting, Feb. 26, 1915. (Gift of Company.)

GOLDFIELD CONSOLIDATED MINES CO. Annual Report, 1914. Goldfield, 1914. (Gift of Company.)

Trade Catalogues

BUTCHART RIFFLE SYSTEM. Bull. No. 1. W. A. Butchart, Denver, Colo.

"CHICAGO PNEUMATIC" STEAM AND POWER DRIVEN COMPRESSORS, CLASS "O."

Bull. 34-M. Chicago Pneumatic Tool Co., Chicago, Ill. March, 1915.

MAKING HEAT PRODUCE. Diamond Power Specialty Co., Detroit, Mich. 16 pp.

SINGLE ROLL COAL CRUSHER. Bull. 141. Jeffrey Mfg. Co., Columbus, Ohio.

AMBLER ASBESTOS CORRUGATED SHEATHING. Keasbey & Mattison Co., Ambler, Pa.

ARTHUR G. MCKEE & CO., CLEVELAND, OHIO. Catalogue No. 4. Describing construction work done by company. 64 pp.

SANFORD-DAY WHEELOGY. Sanford-Day Iron Works, Knoxville, Tenn. March, 1915.

SLUDGE BUCKET. Sparta Iron Works Co., Sparta, Wis. April, 1915.

SPRAY ENGINEERING CO., BOSTON, MASS.

Bull. 101. Sprays for Cooling Condensing Water.

Bull. 151. Washing and Cooling Air for Steam-Turbine Generators.

LABOR SAVER. Stephens-Adamson Mfg. Co., Aurora, Ill. April, 1915.

E. H. STROUD & CO., CHICAGO, ILL.

Bull. 101. Stroud Air-Separation Pulverizer.

Bull. 102A. Stroud Screen Separation, Crushing, etc.

Bull. 103A. Stroud Powdered Coal Burner.

SULLIVAN MACHINERY CO., CHICAGO, ILL.

Bull. 65C. Sullivan Diamond Core Drills. Feb., 1915.

Bull. 71A. Air-Lift Pumping. March, 1915.

VULCAN IRON WORKS, WILKES-BARRE, PA.

Corliss Engines. 30 pp.

Direct-Acting Hoisting Engines. 114 pp. 1910.

Electric Hoists. 64 pp. 1911.

Geared Engines. 34 pp.

Gasoline Locomotives. 20 pp.

Locomotives, "Edition M." 160 pp. 1913.

- "BLASTERS' FRIEND."** New York Blasting Supply Co., 5 Broadway, New York, N. Y. 8 pp. Gives prices and data on blasting supplies and particularly features their circuit detector. This instrument is designed for the purpose of testing electric blasting caps and detonators of all kinds, as well as the blasting wire circuit before firing.
- BALANCES AND WEIGHTS.** Herman Kohlbusch, 172 Broadway, New York, N. Y. 39 pp. Describes and illustrates analytical balances, scales, and weights of precision. Also accessories. Detailed information and prices.
- HARDINGE CONICAL MILL.** Hardinge Conical Mill Co., 52 Broadway, New York, N. Y. 12 pp. Reprint of paper by H. W. Hardinge read before the Canadian Mining Institute on the crushing and dividing of rock and ore. Illustrated. Also reprint of paper read before the American Institute of Mining Engineers on the conical ball-and-pebble mill. Illustrated. General descriptive catalogues on this company's products are included.
- PROSPECTING DRILLS.** New York Engineering Co., 4 Rector St., New York, N. Y. 48 pp. Profusely illustrated and describes their Empire hand drills used in placer mining.
- GOLD DREDGES.** New York Engineering Co., New York, N. Y. 68 pp. Describes and illustrates the Empire dredges and method of construction. Data of interest to gold dredgers are given.
- WAGON LOADERS.** Hudson Machinery Co., Lawyers Bldg., Passaic, N. J. 4 pp. Describes and illustrates the Hudson Duplex wagon loader, giving, as an example of its efficiency, data on an installation by the Lehigh Valley Coal Sales Co. at its plant in South Plainfield, N. J.
- CONVEYOR WEIGHTOMETERS.** Richardson Scale Co., Van Houton Ave., Passaic, N. J. 16 pp. Apparatus for recording weight of material transported on belt and bucket conveyors, cable railways, and overhead transporters. Illustrated. Table of dimensions for standard sizes, and information necessary when inquiries are made relative to installing the machine.
- SAFETY MINE LAMP.** Edison Storage Battery Co., 165 Lakeside Ave., Orange, N. J. 12 pp. Bulletin 1234 for March describes their storage battery lamp for use in mining work, with a comparison of this lamp and others. Illustrated.
- TRANSITS, LEVELS, ENGINEERING INSTRUMENTS.** W. & L. E. Gurley, Troy, N. Y. Several well-illustrated and descriptive catalogues and pamphlets on their instruments, with specifications. Description is given of their transits Nos. 27 and 28 as being particularly adapted for mine surveying.

Notes on Trade Literature

The Harrison Safety Boiler Works of Philadelphia, Pa., has recently issued Engineering Leaflet No. 18, illustrating and describing the testing of V-notch meters, which is a reprint of two papers on the V-notch weir as used in the Cochrane Metering Heater. The earlier paper, by James Barr, describes in detail the apparatus used by him, and the results obtained in tests conducted at Glasgow University in 1907-09, by which he established the fact that with this apparatus the true discharge for given conditions could be determined within one-third of 1 per cent. The later paper, by W. S. Giele, describes a commercial testing apparatus of large capacity installed by the Harrison Safety Boiler Works for the purpose of carrying out an extensive series of tests upon V-notch weirs of different dimensions as actually installed in commercial meter chambers manufactured by them. Both papers are interesting as illustrating the many refinements which are essential in the precise investigation of problems in hydraulics, and the constancy of the V-notch weir when used under known and predetermined conditions.

The Ingersoll-Rand Co., 11 Broadway, New York City, has issued Form 3015, Portable Air Compressors, an illustrated 32-page treatise on the subject of portable air-compressing outfits which have been developed expressly for the contractor, mine operator, factory manager and others employing air tools in connection with work of a temporary or semi-permanent character. A list of bulletins describing in detail each particular line of portable compressors is given, together with catalogues of the various pneumatic tools and equipment mentioned throughout the catalogue.

Form 4032, Jackhamer Mounting, Type JM-6, is a 4-page illustrated bulletin, issued by the same company and describes drill mounting especially adapted to accommodate the Jackhamer, with or without the water feed, for such flat-hole work as drifting and underhand stoping in metal mines where shallow holes are required, for breaking down coal and for driving gangways in coal mines.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Apr. 10 to May 10, 1915:

Members

- AUMAN, WILLIAM, Supt., Susquehanna Coal Co. Lykens, Pa.
 BARTH, CARL G., JR., Min. Engr. 6151 Columbia Ave., Philadelphia, Pa.
 BOOTH, ERIC EDWIN, Min. Engr., Mount Elliott Mine, Selwyn via Cloncurry,
 North Queensland, Australia.
 BOYDELL, HARRY CYRIL, Asst. Mgr., Randfontein Central Gold Min. Co., Box 42,
 Randfontein, Transvaal, South Africa.
 BRADY, SAMUEL HOWARD, Mine Supt. Bohemian Club, San Francisco, Cal.
 BUSH, HARRY EDWARD, Mine Supt., Engels Copper Mining Co. Keddie, Cal.
 CHANDLER, JOHN WINTHROP, Gen. Supt., West End Cons. Min. Co. Tonopah, Nev.
 DICKERMAN, NELSON, Genl. Mgr., The Pato Mines (Colombia), Ltd., Apartado 104,
 Barranquilla, Colombia.
 DOLMAN, SAMUEL GROVE, Min. Engr. Ray, Ariz.
 DYNAN, JOHN LANE, Min. Engr. Tonopah Extension Mining Co., Tonopah, Nev.
 EASLEY, GEORGE ALBERT, Mining. Casilla 27a, La Paz, Bolivia.
 FRANCHOT, DOUGLAS W., Oil Producer. Tulsa, Okla.
 GRIER, CHARLES DENHAM, Cyanide Plant Accountant, Alaska-Treadwell Gold
 Mining Co., Treadwell, Alaska.
 GRUNOW, WILLIAM RANALD, Sampler. Detroit Copper Mining Co., Morenci, Ariz.
 GUGGENHEIM, EDMOND ALFRED, Met. 7 E. 84th St., New York, N. Y.
 HACHITA, M. SHOZO, Chem. and Editor. Lehigh Valley Coal Co., Wilkes-Barre, Pa.
 HALVORSEN, ARTHUR LUDWICK, Chem., Roessler & Hasslacher Chemical Co.,
 Perth Amboy, N. J.
 HARRIS, STEPHEN, Res. Mgr., Hampden Cloncurry Copper Mines, Ltd.,
 Friezland, via Cloncurry, North Queensland, Australia.
 HARRISON, PERRY G., Min. Engr. National Mines Co., National, Nev.
 KEEP, GLENN ALLISON, Supt., Park City Mills. Park City, Utah.
 KNERR, OSCAR ALFRED, Supt. of Mines. Layland, W. Va.
 LINEE, FREDERICK F., Supt., Maryland Steel Co. Sparrows Point, Md.
 MACGOWAN, JOHN K., Director, American Smelting & Refining Co.,
 120 Broadway, New York, N. Y.
 MACKENZIE, KENNETH GERARD, Cons. Chem. The Texas Co., Bayonne, N. J.
 MALLER, JOHN WILLIAM, Min. Engr., Care Cinco Minas Co., Magdalena, Jal., Mexico.
 MITMAN, CARL WEAVER, Mineral Technologist, U. S. National Museum,
 Washington, D. C.
 MORRIS, JOHN THOMAS, Genl. Supt., Weyanoke Coal & Coke Co., Springton, or
 Lowe, W. Va.
 NAGLE, EDWIN BERNARD, Supt., Cuba Copper Co. Santiagode Cuba, Cuba.
 NEWMAN, M. H., Geol. American Zinc, Lead & Smelting Co., Mascot, Tenn.
 PEARSON, JAMES, Mine Mgr. Cottonwood Coal Co., Great Falls, Mont.
 RICHLESEN, WALTER ALFRED, Min. Engr., Constr. Engr., Isle Royal Copper Co.,
 Houghton, Mich.
 ROSSBACH, ERNEST JEROME, Sales Engr. Sullivan Machinery Co., St. Louis, Mo.
 SAYRE, HERBERT A., Coal Operator, Mech. Engr., 1811 College Ave.,
 Des Moines, Iowa.
 SICKLY, ROBERT GLENN, Chem. Buffalo Mines, Ltd., Cobalt, Ont., Canada.
 SMYTH, HAROLD MORGAN, Min. Engr., Asst. Supt., St. Clair Coal Co., St. Clair, Pa.
 SNYDER, PAUL ROBERT, Met. 116 N. Main St., Bethlehem, Pa.
 TIDMORE, JOSEPH H. Pratt City, Ala.
 TRENT, WALTER E., Min. and Met. Engr. Reno, Nev.
 VRANG, CHRISTON MADISEN, Geol. Dawson, Y. T., Canada.
 WALSH, TIMOTHY D., Min. Engr. 130 W. Third Ave., Denver, Colo.
 WARFORD, NORMAN L., Supt., Pulverizing Plant, Anaconda Copper Mining Co.,
 Anaconda, Mont.
 WEBB, CURTIS CHRISTOPHER, Shift Boss, Morococha Mining Co., Morococha,
 Peru.

WHITE, ALFRED G., Mine Economist U. S. Bureau of Mines, Washington, D. C.
 WILBUR, ROLLIN HENRY, Vice-Pres., Lehigh Coal & Navigation Co., Philadelphia, Pa.
 WINSLOW, SIDNEY WILMOT, Pres. United Shoe Machinery Co., Boston, Mass.
 YOUNG, HOWARD I., Mgr., Missouri Mines, Care American Zinc, Lead & Smelting Co.,
 Cartersville, Mo.

Associate Member

LIANG, HUAN TING, Agent, Wah Chang Mining & Smelting Co., 18 Broadway,
 New York, N. Y.

Junior Members

BENHAM, WILLARD MILES, Student Missouri School of Mines, Rolla, Mo.
 CALLAWAY, LAWRENCE A., Student 81 Garfield St., Cambridge, Mass.
 CARLSON, ARTHUR EUGENE 417 Fifth Ave., Ashland, Wis.
 CARPENTER, CLARK BAILEY, Student 1338 Ohio St., Lawrence, Kan.
 CLARKE, JAMES P., JR., Student Little Rock, Ark.
 HODSON, FREDERICK W. 150 Iota Street, Madison, Wis.
 PALLANSCH, ROLLIN A. 14 N. Bassett St., Madison, Wis.
 TSAI, HSIANG, Student Box 735, Golden, Colo.
 WHITE, LEONARD L. 12th and Cheyenne Sts., Golden, Colo.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Apr. 10 to May 10, 1915, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

George Chandler Armstrong, Lawrenceville, Ill.

Proposed by T. T. Read, Anthony F. Lucas, H. D. McCaskey.

Born 1872, Eldora, Iowa. High school education; graduated Marshalltown, Ia., in 1890. Practical laboratory work in experimenting with and actual refining of petroleum and products, having had charge of refineries both in this country and Germany. Also designing and erecting refining plants and apparatus. 1893-1905, Standard Oil Co., Baltimore, Md. 1906-07, Standard Oil Co. of New York, Yokohama, Japan, and Shanghai, China. 1907-08, Deutsch Vacuum Oil Co., Hamburg, Germany. 1908-14, Indian Refining Co.

Present position: General Refinery Supt., Lawrenceville, Ill.

Max Hayden Barber, Nashwauk, Minn.

Proposed by M. F. La Croix, F. G. Rockwell, E. E. White.

Born 1879, Vermontville, Mich. 1898-1903, Univ. of Mich.; B. S. in C. E. 1903-05, Asst. Min. Engr., Ishpeming, Mich.; 1905-07, Min. Engr., Ashland Mine, Ironwood, Mich.; 1907-11, First Asst. Min. Engr., Ishpeming, Mich., Cleveland-Cliffs Iron Co.

Present position: 1911 to date, Supt., Crosby Mine, Cleveland-Cliffs Iron Co.

John A. Bensel, New York, N. Y.

Proposed by W. L. Saunders, E. E. Olcott, C. R. Corning.

Born 1863, New York, N. Y. 1884, Grad., Stevens Institute of Tech. 1891, Member. 1910, Pres., Amer. Soc. of Civil Engineers. 1905, Member, Institution of Civil Engineers (Great Britain). 1884-88, Rodman and Asst. Supervisor, Penn. R. R. Co. 1889-95, Asst. Engr., Dept. of Docks, New York, N. Y. 1896-97, Private practice, Consulting Engr., City of Philadelphia. 1898-1908, Engr. in Chief and Commissioner of Docks, New York, N. Y. 1908-10, Pres., Board of Water Supply, New York, N. Y.

Present position: State Engr. New York State.

Graham Bright, Wilkinsburg, Pa.

Proposed by K. A. Pauly, Kenneth Seaver, H. T. Herr, M. E. Wadsworth.

Born 1876, Pittsburgh, Pa. 1893, Pittsburgh High Schools. 1897, Univ. of Pittsburgh; E. E. 1898-99, Consolidated Traction Co., Pittsburgh. 1899-1902, with J. Bryan, Railway, Industrial and Mining Engineering. 1902-10, Railway Engineering. 1910-15, Min. Dept., Westinghouse Electric & Mfg. Co. 1908-15, Instructor, Electrical Dept., Casino Technical Night School. 1902-13, Associate; 1913-15, Member, Am. Inst. of Electrical Engineers. 1913-15, Member of Committee on Use of Electricity in Mines.

Present position: Engr., Mining Dept., Westinghouse Electric & Mfg. Co.

Charles P. Brooks, Salt Lake City, Utah.

Proposed by W. Fitch, J. M. Callow, Ernest Gayford.

Born 1851, Washingtonville, N. Y. 1870, Grad., Sheffield Scientific School, Yale College. 1870-72, City Engineer's office, New Haven, Conn. 1872-74, Engr., Texas & Pacific Ry. 1874-1915, Office in Salt Lake City, Utah. Engineer for various mining companies, in Utah, Nevada and Idaho.

Present position: Engr., Silver King Coalition Mines Co., Park City, Utah; Chief Consolidated Min. Co. and Grand Central Mining Co., Eureka, Utah.

Daniel J. Carroll, New York, N. Y.

Proposed by W. L. Saunders, George A. Howells, R. S. Carter.

Born 1846, Ireland. For many years engaged in mining and quarrying in Virginia.

Fred Carroll, Sneffels, Colo.

Proposed by Charles A. Chase, Richard A. Parker, George H. Barnhart.

Born 1874, Hiawasee, Ga. 1894, Grad., Colo. State Preparatory School, Boulder. 1894-95, Freshman year. 1896-97, Specialized chemistry, Colo. State Univ., Boulder. 1898, Four months, Nevada, field work, A. C. Campbell, W. L. Stanley, et al. 1899, Three months, southern Utah, field work. 1903, Benton Canon, C. N. Carroll. 1915, Appointed by Governor Carlson Commissioner of Mines of Colo.

Present position: Secretary and Genl. Mgr., Atlas Mining & Milling Co.

Yefah Francis Chen, Pinghsiang Ki, China.

Proposed by S. Ken Huang, Henry S. Drinker, Howard Eckfeldt.

Born 1891, Shanghai, China. 1903-09, St. John's Univ., Shanghai, China. 1909-10, Lawrence Academy, Groton, Mass. 1910-14, Colorado School of Mines, Golden, Colo.; E. M. 1912-13, Summers, visited most of the mining camps in Colorado, Utah, and Montana.

Present position: Asst. Engr., Han-Yeh-Ping Iron & Coal Co., Ltd.

Flavien Aurélien Choffel, Paris, France.

Proposed by W. L. Saunders, George A. Howells, R. S. Carter.

Born 1878, Paris, France. Active member of French Society of Civil Engineers; French Society of Colonial Engineers; French Syndicate of Contractors; French Society of Mining Industry.

Present position: 1902 to date, Mgr., Ingersoll-Rand Co.

Benjamin Franklin Cresson, Jr., Montclair, N. J.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1873, Philadelphia, Pa. 1883-90, Academy of the Protestant Episcopal Church, Philadelphia. 1890-93, Lehigh University. 1893-94, Univ. of Pennsylvania; B. S. Member, American Society of Civil Engineers; Institution of Civil Engineers of Great Britain; International Congresses of Navigation; Municipal Engineers of New York; Member and Director of the American Association of Port Authorities. 1894-97, Chainman, rodman, draftsman, Lehigh Valley Railroad Co. 1897-99, Instrument man, Pennsylvania Ave. subway and tunnel work in Philadelphia. 1899-1900, Asst. Engr., West Virginia Short Line Railroad. 1900-01, Asst., Jacobs & Davies, New York, on New York tunnel projects, Hudson tunnels. 1901, Asst. Engr., Atlantic Ave. improvements, Brooklyn. 1901-10, Asst. Engr., Engineer of Alignment and Resident Engr., Pennsylvania Railroad Tunnels and Terminals into New York—North River Division. 1910-13, First Deputy Commissioner, Department of Docks and Ferries, New York, N. Y.

Present position: Chief Engr., New Jersey Harbor Commission.

William Barry Daly, Butte, Mont.

Proposed by John Gillie, B. H. Dunshee, C. W. Goodale.

Born 1873, California. 1879-1890, Public schools. 1890-91, Taught school. 1892-1894, Studied law. 1894, Admitted to practice law by California Supreme Court. 1894-99, Practiced law in San Francisco, Cal. 1899-1900, Mine timekeeper. 1900-04, Underground miner. 1905-06, Supt.'s clerk and chief timekeeper. 1907-10, Foreman, East and West Gray Rock mines. 1910-12, Foreman, Bell-Diamond and Gray Rock mines. 1913, Supt. Employment Dept.; 1914, Efficiency Engr., Anaconda Copper Min. Co.

Present position: 1914 to date, Genl. Supt. of Mines, Anaconda Copper Min. Co.

William Jefferson Deavitt, Santa Eulalia, Chih., Mexico.

Proposed by S. F. Shaw, Basil Prescott, John M. Brooks, Jr.

Born 1883, Montpelier, Vt. 1888-1900, Public schools, Montpelier, Vt. 1900-04, Williams College; A. B. 1904-06, Mass. Institute of Technology; B. S. 1906-07, Canadian Copper Co., Copper Cliff, Ont. 1907, Engr., Munro Iron Min. Co., Iron River, Mich. 1907-11, Assayer, Engr., Foreman, etc., Minas Tecolotes y Anexas, Santa Barbara, Chih., Mexico. 1911, Cashier and Surveyor, Reforms Unit, M. Guggenheim Sons, Cuatro Ciénegas, Coah., Mexico. 1911-12, Supt., Minas Veta Grande y Anexas, Villa Escobedo, Chih., Mexico.

Present position: Supt., Santa Eulalia Unit, American Smelting & Refining Co.

Hallard W. Foester, Esqueda, Son., Mexico.

Proposed by L. R. Budrow, R. T. Mishler, Charles M. Heron.

Born 1891, Harney, Ore. 1909, Nampa High School, Idaho. 1913, Univ. of Idaho; B. S. 1909-10, Summers, Golden Crown Min. Co., Owyhee Co., Ida. 1911, Summer, Reclamation Service on the Boise-Payette Project, Ida. 1912, Summer, assayer for lessees, Trade Dollar Consolidated Property, Silver City, Ida.

Present position: 1913 to date, Mill Shift Boss, Lucky Tiger Min. Co.

Sheppard B. Gordy, Rancagua, Chile.

Proposed by J. F. McClelland, Arthur F. Taggart, L. W. Bahney.

Born 1889, Ansonia, Conn. 1907, Public schools. 1907-10, Ph. B.; 1910-12, E. M., Sheffield Scientific School. 1909, Mucker, Empire Steel & Iron Co., Mt. Hope, N. J. 1910, Surveyor, N. Y., N. H. & H. R. R. 1911, Summer Student, Pewabic Mine, Mich.

Present position: Genl. Mine Foreman, Braden Copper Co.

Wilbur W. Graff, Ishpeming, Mich.

Proposed by J. E. Jopling, Carl Brewer, E. E. White.

Born 1877, Rushville, Ill. 1901, Lehigh Univ.; E. M. 1901-04, Asst. Min. Engr., Cleveland-Cliffs Iron Co. 1904-09, Supt., Cliffs Shaft and Moro Mines, Cleveland-Cliffs Iron Co.

Present position: 1904 to date, Supt., North Lake District, Cleveland-Cliffs Iron Co.

Ernest R. Graham, Terry, S. D.

Proposed by Charles A. Chase, Lee D. Dougan, John V. N. Dorr.

Born 1885, Crosswell, Mich. 1891-1902, Public schools, Crosswell, Mich. 1902-04, Michigan Agricultural College, Lansing, Mich. 1905-07, Michigan College of Mines, Houghton, Mich.; B. S. and E. M. 1904-05, Mining, Victoria Copper Co., Victoria, Mich. 1906-07, Prospecting near Cobalt, Ont. 1907-08, Michigan Gold Mining Co., Custer, S. D. 1908-09, Laborer, Mogul Min. Co., Pluma, S. D. 1909-10, Michigan & Montana Development Co., Wickes, Mont. 1911, Miner, Butte & Superior Copper Co., Butte, Mont.; miner, Tramway Mine, Anaconda Copper Co., Butte, Mont.

Present position: Supt., Mogul Mining Co.

Daniel Guggenheim, New York, N. Y.

Proposed by Willard S. Morse, Judd Stewart, A. Eilers.

Born 1856, Philadelphia, Pa. Philadelphia public and high schools. Also studied abroad.

Present position: Pres., Guggenheim Exploration Co., American Smelting & Refining Co., American Smelters Securities Co., Chile Copper Co., Chile Exploration Co., and have material interests in and official connection with many mining companies.

John Herman, Los Angeles, Cal.

Proposed by C. Colcock Jones, S. W. Mudd, F. J. H. Merrill.

Born 1878, Wilber, Neb. 1900, Univ. of Neb.; B. S. 1900-01, Assay office, Canon City, Colo. 1901-02, Assayer, Copper King Min. Co., Canon City, Colo. 1902, United Verde in charge of generating CO₂ for mine fires. 1903-04, Assay office, Bisbee, Ariz. 1905-06, Supt., Old Keystone Min. Co., Globe, Ariz. 1906-07, Research chemist, Cananea Consolidated Min. Co., also charge of reverberatory furnace.

Present position: 1907 to date, Laboratory, Los Angeles, Cal., ore testing and chemical works

Stacy H. Hill, Duluth, Minn.

Proposed by William G. LaRue, K. M. Way, Edwin J. Collins.

Born 1888, Moline, Ill. 1902-04, Todd Seminary, Woodstock, Ill. 1905-07, Denison Univ., Granville, O. 1907-09, Michigan College of Mines, Houghton, Mich. 1909-11, Salesman, Ingersoll-Rand Co., Houghton, Mich.

Present position: 1911 to date, Mgr., Ingersoll-Rand Co.

John E. Hodge, Minneapolis, Minn.

Proposed by C. K. Leith, E. C. Holden, E. J. Longyear.

Born 1862, Redford, Wayne Co., Mich. 1884, Grad. Ypsilanti Seminary. 1888, Grad., Univ. of Michigan; B. S. 1902, Exploring engineering work, E. J. Longyear, Hibbing, Minn. 1903-11, Longyear & Hodge.

Present position: Vice-Pres., E. J. Longyear Co.

Edmund Townsend Lednum, Chicago, Ill.

Proposed by L. B. Miller, Charles F. Rand, Luther V. Rice.

Born 1878, Mount Pleasant, Del. Middletown (Del.) Academy. Until 1896, Wilmington (Del.) High School. Technical education from study and observation. 1896-1901, Asst. to G. F. Knapp, Eastern Manager (Phila., Pa.), Oglebay, Norton & Co., Cleveland, O., producers and sales agents for iron ore. 1901, Crocker Bros. (N. Y.), pig iron. 1902-04, Supt. in charge of construction on Penn. R. R., Smith Const. Co. (Phila. Pa.), R. R. contractors.

Present position: 1904 to date, Explosives, E. I. DuPont de Nemours Powder Co.

William Dixon Leonard, Garfield, Utah.

Proposed by P. A. Mosman, E. P. Mathewson, H. A. Proseer, W. H. Howard, W. S. Morse.

Born 1864, Denver, Colo. Public schools of Illinois and Colorado and private instruction in chemistry and metallurgy. 1883-90, Assayer, Arkansas Valley Smelting Co. 1890, Assayer, Philadelphia Smelter. 1890-95, Assayer, M. Guggenheim's Sons. 1895-1901, Assayer, Philadelphia Smelter. 1901-06, Supt. and Asst. Supt., Murray Plant, American Smelting & Refining Co.

Present position: 1906 to date, Supt., Garfield Smelter.

Harley Martin Lindsey, Ray, Ariz.

Proposed by C. E. Addams, W. S. Boyd, L. S. Cates.

Born 1889, McGregor, Tex. High school. 1910-11, Underground foreman, DeBeers Consolidated Mines, Ltd., Kimberley, So. Africa. 1911-12, Voorspoed Diamond Min. Co., So. Africa.

Present position: 1913 to date, Shift Boss, Ray Consolidated Copper Co.

Frederick J. MacIsaac, New York, N. Y.

Proposed by W. L. Saunders, R. S. Carter, George A. Howells.

Born 1863, Waterloo, Iowa. Public schools and private instructor. 1886-1901, FitzSimons & Connell Co., Contractors, Chicago, construction of tunnels, bridges, etc. 1901-06, Contracting on tunnels, shafts, etc. 1906-08, Winston Bros. Co., Minneapolis, on St. Paul Pass tunnel, Bitter Root Mts.

Present position: 1908 to date, Contractor, Grant Smith & Co., on Kelso tunnel, Seattle tunnel, and Catskill aqueduct.

Austin Gerry Marsh, McGill, Nev.

Proposed by C. B. Lakenan, H. A. Guess, P. A. Mosman.

Born 1887, Carnegie, Pa. 1898-1904, Public schools, Pueblo, Colo. 1905-06, Horace Mann School, New York, N. Y. 1906-10, Columbia Univ., New York; E. M. 1906-10, Summer work in assay office and laboratories of Eilers and Pueblo Smelters, American Smelting & Refining Co. 1910-13, Operator, sampler, repair man, floor boss, draftsman, Nevada Consolidated Copper Co.'s concentrator (formerly Steptoe Valley Smelting & Mining Co.), McGill, Nev.

Present position: 1913 to date, Metallurgist, Concentrator Department, Nevada Consolidated Copper Co.

John Douglas Martin, East St. Louis, Ill.

Proposed by Elias S. Gatch, Kennett Barnes, Philip N. Moore.

Born 1881, Little Rock, Ark. 1902, Grad., St. Louis Manual Training School. 1902-03, Foreman, R. R. construction in Oklahoma, Fruin-Bambrick Construction Co. 1903-04, Foreman, street construction in St. Louis, Granite Bituminous Construction Co. 1904-05, Instrument work, engineering party, St. Louis & San Francisco Railroad Co., Arkansas and Louisiana. 1905-07, Draftsman and instrument work, engineering party, Oregon Railroad & Navigation Co. 1907-12, Engr., construction work, Granby Mining & Smelting Co., Neodesha, Kan.

Present position: 1912 to date, Asst. Supt., Rose Lake Smelter, Granby Mining & Smelting Co.

William F. Mellen, Newark, N. J.

Proposed by Richard C. Patterson, Jr., C. A. Burdick, Sidney L. Wise.

Born 1890, New Orleans, La. New Orleans Military Academy. 1904-05, Spring Hill College, Mobile, Ala. 1909-12, Supt., United Aluminum Ingot Co., Newark, N. J. 1912-14, Pres., Newark Smelting & Refining Co. 1914-15, Consulting Metallurgist.

Present position: Experimental Engr., Continuous Casting Corporation.

Howard E. Perry, Terlingua, Tex.

Proposed by Walter E. Koch, F. H. Fovargue, Richard Tarantous.

Born 1860, Cleveland, Ohio.

Present position: Pres., Chisos Mining Co.

Edson S. Pettis, Mill Valley, Cal.

Proposed by Abbot A. Hanks, Louis D. Mills, Sidney E. Bretherton.

Born 1887, Santa Cruz, Cal. Public schools, Oakland and Mill Valley, Cal. 1904, Assay Student, Hanks Laboratory, San Francisco. 1905, Apprentice in ore testing plant. 1908, Van der Naillen School of Engineering, Oakland, Cal. 1906, Supt., Cyanide Plant, Combination Mill, Nev. 1907, Construction work, Montana Tonopah Mill, Nev.; Ore testing, Asst. to Francis L. Bosqui. 1908-11, Short engagements, mill construction, operation, special assaying, in Nevada, Arizona, California, and Mexico. 1912, Supt., Mascot Mill, Ariz.

Present position: 1913 to date, Supt., California Ore Testing Co., San Francisco, Cal.

James A. Potter, Ajo, Ariz.

Proposed by L. D. Ricketts, F. L. Antisell, J. C. Greenway.

Born 1879, Bay City, Mich. 1898-1900, Univ. of Mich. 1900-02, Michigan College of Mines. 1902-06, Chem. and Engr., Oliver Min. Co. 1906-13, Field engr. and draftsman, Calumet & Arizona Min. Co.

Present position: 1913 to date, Supt. Leaching Plant, New Cornelia Copper Co.

Hugh M. Roberts, Minneapolis, Minn.

Proposed by Beriah Magoffin, Jr., Frank G. Jewett, E. J. Longyear.

Born 1885, Superior, Wis. 1890-1902, Public schools; Nelson Dewey High School, Superior, Wis. 1907-10, Univ. of Wisconsin, mathematics and geology. 1902-03, Laborer, Ry. Construction. 1902, Chairman, A. P. Silliman, Hibbing, Minn. 1904-06, Rodman and Instrumentman, Duluth, Missabe & Northern Ry., Duluth, Minn. 1906-07, Engr. and foreman, Oliver Iron Min. Co., Coleraine, Minn. 1909-10, Genl. Supt. and Asst. to Dr. C. K. Leith, Hudson Bay, Canada. 1910-11, Asst. to Dr. C. K. Leith, Brazil.

Present position: 1911 to date, Field Geol., E. J. Longyear Co.

Eugene Roger, Elisabethville, Congo Belge, W. Africa.

Proposed by A. E. Wheeler, Frederick W. Snow, Thomas T. Read.

Born 1885, Bethune, France. 1904-08, Institut industriel du Nord de la France, Lille. 1908-13, Chief chemist, Compagnie française des mines de Bor, Servia.

Present position: 1913 to date, Asst. Met., Union Minière du Haut Katanga.

Herbert S. Salmon, Bessemer, Ala.

Proposed by George G. Crawford, Frank H. Crockard, C. E. Abbott.

Born 1887, Frankfort, Ind. 1901-05, Frankfort High School. 1905-07, Purdue University. 1908-10, International Correspondence Schools. 1906, Draftsman, Geo. L. Meaker, Arch. Iron Works, Evansville, Ind. 1907, Levelman, Evansville & Terre Haute R. R. Co. 1907, Draftsman, L. & N. R. R. Co., Birmingham, Ala. 1908-12, Division Engr., T. C., I. & R. R. Co. Ore Mines, Bessemer, Ala.

Present position: 1912 to date, Mine Inspector, T. C., I. & R. R. Co., Ore Mines.

Wilfred Sykes, Pittsburgh, Pa.

Proposed by C. F. W. Rys, Hugh P. Tiemann, J. S. Unger.

Born 1879, Palmerston, North New Zealand. 1886-96, State and private schools, Victoria; high school, Melbourne. 1896-1900, Electrical and mechanical engineering, Melbourne Technical College; and Elect. Engrg., Melbourne Univ. 1900-03, Asst. Engr., Knox Schlapp & Co., Melbourne, designing electrical and mining installations. 1903, Mgr., branch office Knox Schlapp & Co., Sydney. 1903-05, Engr. in charge electrical work of above. 1905-06, Elect. Engr., Staerker & Fischer, in charge of offices in Victoria, South Australia and Tasmania on transfer of business from Knox Schlapp & Co. 1907-09, Engr., Allgemeine Elektrizitäts Gesellschaft, Berlin, handling mining work for So. Africa.

Present position: 1909 to date, Westinghouse Elect. & Mfg. Co., East Pittsburgh, in charge Industrial Engineering, which includes all mining work. Chairman of Com. on the Use of Electricity in Mines of Amer. Inst. E. E.

John Roy Theonen, Garson, Ont., Canada.

Proposed by John T. Fuller, E. T. Corkill, F. W. McNair.

Born 1886, Ithaca, Mich. 1894-1904, Grad. grammar and high school, Sault Ste. Marie, Mich. 1905-07, Mechanical Course Michigan Agricultural College, Lansing, Mich. 1907, Michigan College of Mines, Houghton, Mich. 1904-05, Contractors, MacArthur Bros. Co. and Grant Smith Co. 1908, Traveling, Central Michigan Film Co. 1909, Surveying, Clark & Lounsberry, Code & Code, Cobalt and Gowganda. Part time prospecting. Foreman, Lucky Godfrey Min. Co. 1910-13, Time checker to Mine Supt., Canadian Copper Co.

Present position: 1914 to date, Mine Supt. and Safety Engr., Mond Nickel Co., Ltd.

Henry Adolph Tobelmann, Ajo, Pima Co., Ariz.

Proposed by L. D. Ricketts, A. G. McGregor, F. L. Antisell, J. C. Greenway.

Born 1876, Newark, N. J. Newark public schools. 1895-96, Bethlehem Prep. School. 1896-1900, Lehigh Univ.; B. S. in Metallurgy. 1892-95, Dr. Lucius Pitkin, New York City. 1900-01, Bethlehem Steel Co. 1901, Dexter Portland Cement Co. 1901-02, Baltimore Copper R. & S. Co. 1902-04, Old Dominion Copper M. & S. Co. 1904-06, U. S. G. S.-U. S. R. S., Roosevelt, Ariz. 1906-10, C. & A. Mining Co., Bisbee, Ariz. 1910-14, Douglas, Ariz.

Present position: Chemist, C. & A. Mining Co.

Albert Louis Toenges, Rivermines, Mo.

Proposed by George B. Rodgers, O. M. Bilharz, E. J. Rossbach.

Born 1889, Dayton, O. 1896-1903, Grammar school, Dayton, O. 1903-07, High school, Dayton, O. 1908-12, Colorado School of Mines, Golden, Colo.; E. M. 1912, Colorado Fuel & Iron Co., Tabasco, Colo. 1912-14, Engr., Oliver Iron Mining Co., Hibbing, Minn. 1914, Engr., Duluth, South Shore & Atlantic R. R., Duluth, Minn.

Present position: Engr., Doe Run Lead Co.

William Frederick Vansant, Victor, Colo.

Proposed by F. K. Brunton, Howland Bancroft, D. W. Brunton.

Born 1883, Clifton, Ill. 1900, High school. 1912, Course Min. Engrg., Int. Correspondence Schools. 1906-08, General man, mining, Woods Inv. Co., Victor, Colo.

Present position: Pres. and Genl. Mgr., Copeland Ore Sampling Co.

Associate Members

Ernest Robinson Ackerman, Plainfield, N. J.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1863, New York, N. Y. Member, Engineers' Club. Was Acting Governor of N. J. under Woodrow Wilson.

Present position: Pres., Lawrence Cement Co.

Arthur A. Bonsack, St. Louis, Mo.

Proposed by W. L. Saunders, H. A. Wheeler, Philip N. Moore.

Born 1875, St. Louis, Mo. St. Louis High School. Ingersoll-Rand Co., St. Louis and Chicago, for past 18 years.

Present position: Mgr., Ingersoll-Rand Co.

Frank L. Connable, Wilmington, Del.

Proposed by Joseph T. Hilles, William W. Hearne, Frank C. Roberts.

Born 1877, Xenia, Ohio. M. I. T., Boston, Mass., and prep. school. 1893, Pres., Chattanooga Powder Co., Chattanooga, Tenn.; also identified with many other corporations here.

Present position: Vice-Pres., E. I. duPont de Nemours Powder Co.

George Thomas Cousins, New Rochelle, N. Y.

Proposed by W. L. Saunders, George A. Howells, R. S. Carter.

Born 1869, Brooklyn, N. Y. Public schools, Norwalk, Conn. 1887, Norwalk Iron Works, Norwalk, Conn.; apprentice machinist. 1897, Draftsman, Hartford Cycle Co., Hartford, Conn. 1898, Draftsman, C. & C. Electric Co., Garwood, N. J. 1898, Draftsman and Engr., Ingersoll-Rand Co., Easton, Pa. 1904, Engr. and salesman, Ingersoll-Rand Co., New York. 1906, Mgr. and Engr., Ingersoll-Rand Co., Butte, Mont. 1909, Mgr. and Engr., El Paso, Tex. 1910, Mgr. and Engr., Ingersoll-Rand Co., Johannesburg, So. Africa.

Present position: Engr., Rock Drill Dept., Ingersoll-Rand Co.

William Russell Grace, Long Island, N. Y.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1878. Builder of mining machinery.

Present position: Vice-President, Ingersoll-Rand Co.

Wah S. Lee, Stanford University, Cal.

Proposed by C. F. Tolman, Jr., D. M. Folsom, James C. Ray.

Born 1892, San Francisco, Cal. 1915, Stanford Univ.; A. B. Work done in Department of Geology with special work in ore deposits.

Present position: Unemployed.

Joseph Leiter, Washington, D. C.

Proposed by E. H. Gary, John H. Hammond, Alexander C. Humphreys, Robert W. Hunt, J. Parke Channing.

Born 1868, Chicago, Ill. 1991, Harvard; A. B. 1898-1902, Pres. and Genl. Mgr., Zeigler Coal Co., Zeigler, Ill.

Present position: Pres., Zeigler Coal Co.

John William Straney, Rancagua, Chile.

Proposed by Pope Yeatman, B. T. Colley, R. K. Stockwell.

Born 1890, Pittsburgh, Pa. 1909-1911, Surveyor for R. W. Nesbit, Municipal Engr., Pittsburgh, Pa. 1911-14, Chief Surveyor, Pittsburgh Crucible Steel Co., Midland, Pa.

Present position: 1914 to date, Civil Engr., Braden Copper Co.

Junior Members

Herbert Arthur Bedworth, New Haven, Conn.

Proposed by L. W. Bahney, Arthur F. Taggart, J. F. McClelland.

Born 1892, Syracuse, N. Y.

Present position: Student, Metallurgical Engineering, Sheffield Scientific School.

Charles Winegar Crispell, New Haven, Conn.

Proposed by L. W. Bahney, J. L. McClelland, Arthur F. Taggart.

Born 1894, Kingston, N. Y.

Present position: Senior, Mining Engineering Department, Sheffield Scientific School.

James Lawrence Head, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, Durward Copeland.

Born 1894, Detroit, Mich. Public schools, Peru, Ind.; Moberly, Mo. 1912, Grad., Moberly High School, Moberly Mo. 1912, Summer, Yard Clerk, Wabash R. R., Moberly, Mo. 1913, summer, Silver Spoon Mine, Leadville, Colo. 1914, Summer, in care of S. T. McDonald, City Engineer, Moberly, Mo.

Present position: 1912 to date, Student, Missouri School of Mines and Metallurgy.

Lloyd Hoffman, So. Bethlehem, Pa.

Proposed by F. Lynwood Garrison, Howard Eckfeldt, Joseph W. Richards.

Born 1887, Middle Valley, N. J. 1907, Grad., Peddie Institute. 1907-09, Lehigh Univ., mining course; 1913-15, Geology course. 1909, King Edward mine, Cobalt, Ont. 1909-13, Prospecting, Northern Ontario. 1913, With F. L. Garrison, H. L. Mead, W. R. Cranston, and W. Henderson Scott, in Colombia.

Present position: Senior student in Lehigh Univ.

Henry W. Kaanta, Golden, Colo.

Proposed by F. W. Traphagen, W. G. Haldane, Harry J. Wolf.

Born 1889, Rock Springs, Wyo. 1908, Grad., Telluride High School, Colo. 1908-09, Colorado School of Mines. 1907-08, Accountant, Black Bear Mining Co., Telluride, Colo. 1909-10, Kaanta-Nevada Leasing Co., Telluride, Colo. 1910-12, Director, Sec.-Treas., Black Bear Mining Co., Telluride, Colo.

Present position: Student, Colorado School of Mines.

Paul Raymond Murphy, Madison, Wis.

Proposed by Richard S. McCaffery, C. K. Leith, Mack C. Lake.

Born 1894, Benton, Wis. 1910-11, State Normal School, Platteville, Wis. 1912-14, Wis. State Mining Trade School. 1912, Summer, Installation of mining machinery and engineer of plant, Beloit Elmo Min. Co., Platteville, Wis. 1913, Fireman, Oliver Min. Co. Mesabi Range and mines; Lucky Twelve Mine, Benton, Wis. 1913, Surveying of mines, land and city. 1913-14, Highway Inspector and County Engr., Allamakee Co., Iowa.

Present position: Student, Univ. of Wisconsin.

William Walter Taylor, New Haven, Conn.

Proposed by Arthur F. Taggart, L. W. Bahney, J. F. McClelland.

Born 1888, Birmingham, England. 1901-05, Hillhouse High School, New Haven. 1905-08, Sheffield Scientific School; Ph. B. 1909, Experimental mill, Ray Cons. Copper Co., Kelvin, Ariz. 1909-11, Engr. on construction, Ray Cons. Copper Co., Hayden, Ariz. 1911-12, Dome Mines Co., Ltd., So. Porcupine, Ont., Canada. 1912-14, Canadian Copper Co., Frood Mine and Copper Cliff, Ontario.

Present position: Student, Sheffield Scientific School.

Chuang Yen, Pittsburgh, Pa.

Proposed by Ying-Chieh Kuang, Roswell H. Johnson, H. B. Meller.

Born 1890, Shensi, China. 1907-10, Shanghai High School. 1910-13, National Univ., Peking. 1913, Columbia Univ.

Present position: Student, University of Pittsburgh.

For Reinstatement

Frank Harold Brown, Elizabeth, N. J.

Member, 1905 to 1910. Since former membership, 1910 to date, Director and Consulting Engr., Durango Min. & Mill. & Explor. Co., San Lucas, Durango, Mexico. In partnership with Charles Fowles, property at Yerbabuena, Durango, Mexico. Director and Cons. Engr., New Utah Bingham Mining Co., Bingham, Utah.

Present position: Consulting Engr. and Director, Federal Company, Inc.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Apr. 10 to May 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 and 101, April and May, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of May 10, 1915.

ADAMS, HUNTINGTON.....Care W. R. Grace & Co., Iquique, Chile.

ALEXANDER, CURTIS, R. F. D. 2, Box 308-11, 4508 Terrace Drive,
Kensington Park, San Diego, Cal.

ANDERSON, ALEXANDER.....27 Inverleith Row, Edinburgh, Scotland.

ANDERSON, ANDREW P.....1504 Hobart Bldg., San Francisco, Cal.

ANDERSON, CARL N.....Care Alaska Consolidated Copper Co., Strelina, Alaska.

- BARNUM, W. E. 821 E. Fourth St., Tucson, Ariz.
 BERRY, EDWIN S., Care American Smelting & Refining Co., 120 Broadway,
 New York, N. Y.
 BOHN, CHARLES A., Care American Smelting & Refining Co., 120 Broadway,
 New York, N. Y.
 BOORAEM, ROBERT E. 72 Brookside Ave., Ridgewood, N. J.
 BOYLE, CLARENCE, JR., Salesman, Clarence Boyle, Inc., 1211 Lumber
 Exchange Bldg., Chicago, Ill.
 BRANDT, M. S., Chief Engr. and Gen. Mgr., Seneca Cons. Gold Mines, Seneca,
 Plumas Co., Cal.
 BREINL, JOSEPH C. Instructed to hold all mail.
 BRITT, RICHARD H. 29 Firglade Ave., Springfield, Mass.
 BROWN, THOMAS J., Care Nova Scotia Steel & Coal Co., Ltd., Sydney Mines,
 N. S., Canada.
 CALDWELL, FOREST B., Supt., Mexican Candelaria Co., S. A., San Dimas,
 Dur., Mexico.
 CHENNEL, J. A. Instructed to hold all mail.
 CLARK, C. DAWES. Room 1802, 120 Broadway, New York, N. Y.
 CLARKE, SIMON S. Care Doe Run Lead Co., Rivermines, Mo.
 COLLINS, HENRY F., Huelva Copper & Sulphur Mines, Ltd., Cueva de la Mora,
 Valdelamusa, Huelva, Spain.
 CONOVER, M. J. Care Montana Club, Tonopah, Nev.
 COOK, PAUL R. 4159 Grand Boulevard, Chicago, Ill.
 COOKE, HENRY M. A., Ooregum Gold Min. Co. of India, Ltd., Oorgaum,
 Prov. of Mysore, India.
 CORNELL, R. T. 80 Maiden Lane, New York, N. Y.
 CRAMPTON, THEODORE H. M. Goodsprings, Nev.
 CRANDALL, RODERIC. 604 Tennyson Ave., Palo Alto, Cal.
 CREWE, L. C., Care Aetna Explosives Co., Inc., 2 Rector St., New York, N. Y.
 CUSHING, DANIEL. 1874 E. 81st St., Cleveland, Ohio.
 DENMAN, HEBER. Care Bache-Denman Coal Co., Ft. Smith, Ark.
 DINGWALL, W. B. A. P. O. Box 179, San Antonio, Tex.
 D'INVILLIERS, EDWARD V. Geol. and Min Engr., 518 Walnut St., Philadelphia, Pa.
 DORR, J. V. N. Room 2843, Whitehall Bldg., New York, N. Y.
 DOUGLAS, THEODORE. 245 W. 55th St., New York, N. Y.
 DUNSTER, CARL B. Care Breitung & Co., Ltd., 11 Pine St., New York, N. Y.
 DYE, A. V. Care Phelps, Dodge & Co., Bisbee, Ariz.
 ENOS, H. C. Hotel Paso del Norte, El Paso, Tex.
 EPPLEY, MARION. Princeton, N. J.
 ESEBLYSTYN, J. N. Hillman Land & Iron Co., Hematite, Trigg Co., Ky.
 ESTABROOK, E. L. Care Standard Oil Co. of New York, Peking, China.
 EVANS, GEORGE WATKIN. 423-24 New York Block, Seattle, Wash.
 FACKENTHAL, B. F., JR. Riegelsville, Pa.
 FELLOWS, AUBREY. Salisbury, Mo.
 FISHER, HOWELL T. P. O. Box 411, Englewood, N. J.
 FLYNN, F. N., Smelter Supt. Arizona Copper Co., Ltd., Clifton, Ariz.
 FORMAN, J. K. Box 1105, Collinsville, Okla.
 FREEMAN, R. R., JR. 40 Grand View Ave., Wollaston, Mass.
 FYFE, ALEXANDER, Care Socorro Gold & Silver Mine, Ltd., Valle de Angeles,
 Tegucigalpa, Honduras.
 GABY, WALTER E., Care Phi Delta Theta Fraternity, 585 W. 113th Street,
 New York, N. Y.
 GARFIELD, A. S. 67 Ave. de Malakoff, Paris, France.
 GILLET, LORENZO M. 25 W. Tenth St., New York, N. Y.
 GILMORE, LUTHER E., Care Baltimore Malleable Iron & Steel Castings Co.,
 2621 N. Charles St., Baltimore, Md.
 GITTENS, A. W. No. 2 Iris Apts., Park View Ave., Pittsburgh, Pa.
 GOLDSTEIN, D. E. Care Bachman House, Hazleton, Pa.
 GRAMMEL, PAUL, Director, Mynbouw Maatschappy Aequator, Payacombo,
 Dutch East Indies.
 GREYER, W. S. Greenfield, Mo.
 GREY, GEORGE R., 58 Princes St., Troyeville, Johannesburg, Transvaal, South Africa.
 GRIFFIN, FITZ ROY N. Instructed to hold all mail.
 GUESS, H. A., Care American Smelting & Refining Co., 120 Broadway, New York, N. Y.
 GUITERMAN, FRANKLIN, Care American Smelting & Refining Co., 120 Broadway,
 New York, N. Y.

HALL, JOHN H.	Care Taylor-Wharton Iron & Steel Co., High Bridge, N. J.
HAMILTON, E. H.	186 Boulevard, Port Norfolk, Va.
HASE, H. C.	Care Cusi Mining Co., Cusihiuriachic, Chih., Mexico.
HAWKHURST, ROBERT, JR.	2705 Buchanan St., San Francisco, Cal.
HENDERSON, H. P.	60 Broadway, New York, N. Y.
HIESTER, ARTHUR J.	1026 St. Paul St., Denver, Colo.
HOFFMANN, ROSS B.	2 Rector St., New York, N. Y.
HOLLIS, HENRY L.	1025—122 S. Michigan Ave., Chicago, Ill.
HOLME, P. E.	El Oro, Mex., Mexico.
HURSH, ROBERT, Care Empire Zinc Co., Apartado 214,	Monterrey, N. L., Mexico.
IMHOFF, ALEXANDER	1020 Colton St., Los Angeles, Cal.
INGLIS, J. F.	Care Wasatch Utah Mine, R. D. 4, Sandy, Utah.
JACKSON, F. H., Care El Rayo Mining & Developing Co.,	714 Mills Bldg., El Paso, Tex.
JONES, J. ELMER, Supt.	Mill Creek Coal Co., Hazleton, Pa.
KEENEY, ROBERT M.	Somersville, Conn.
KEFFER, FREDERIC, Min. Engr.	214 Hutton Bldg., Spokane, Wash.
KELLEY, ARTHUR L.	P. O. Box 12, Pearce, Ariz.
KENNEDY, GEORGE A.	2741 Boulevard F, Denver, Colo.
KICHLINE, F. O.	110 S. First Ave., Bethlehem, Pa.
KLOPSTOCK, PAUL	40 Wall Street, New York, N. Y.
KOZMINSKY, LESLIE M., The "Grove" (Cootamundra), Ltd.,	Cootamundra, N. S. W., Australia.
LATHE, FRANK E.	Box 26, Grand Forks, B. C., Canada.
LATHROP, L. J.	2174 39th Ave., Oakland, Cal.
LAURIE, FRANK C.	Care C. E. Hendrickson, 331 Norton St., New Haven, Conn.
LEA, JAMES	P. O. Box 50, Nigel, Transvaal, South Africa.
LEACH, A. A., JR.	Box 253, Santa Rita, N. M.
LINN, W. J.	2629 Michigan Ave., Chicago, Ill.
M'ALLEN, J. L., Engr.	Doheny & Thomson, Montreal, Canada.
MCCORMICK, HENRY, JR.	P. O. Box 548, Harrisburg, Pa.
MCINTOSH, F. K.	Hurley, Wis.
McMILLEN, R. H.	331 Second St., Aspinwall, Pa.
MACDONALD, JESSE J.	Care Ray Cons. Copper Min. Co., Hayden, Ariz.
MADGE, WILLIAM C.	7 Gracechurch St., London, E. C., England.
MASTERS, J. E.	627 N. Fourth Ave., Tucson, Ariz.
METCALF, FRANK A.	P. O. Box 916, Juneau, Alaska.
METZGER, WILLIAM G.	3122 Sheridan Rd., Chicago, Ill.
MILLER, HARRY HUNTINGTON	Instructed to hold all mail
MISHLER, R. T., El Tigre Mining Co., S. A., Esqueda, Son., Mexico,	via Douglas, Ariz.
MOCINE, JOHN	1109 Merchants National Bank Bldg., San Francisco, Cal.
MOORE, EDWARD W.	Care J. F. Gilchrist, 5406 Blackstone Ave., Chicago, Ill.
MORRISON, H. A.	Care Aurora Consolidated Mines Co., Aurora, Nev.
MORSE, WILLARD S., Care American Smelting & Refining Co.,	120 Broadway, New York, N. Y.
MOSES, PERCIVAL SNEED, Sampler	Butte & Superior Copper Co., Butte, Mont.
MOSMAN, P. A., Care American Smelting & Refining Co.,	120 Broadway, New York, N. Y.
MURPHY, R. E.	Juneau, Alaska.
NACK, CHARLES	800 W. 18th St., Los Angeles, Cal.
NEUSTAEDTER, A.	Mineral, Va.
NOON, THOMAS F.	Illinois Zinc Co., Peru, Ill.
NORDENHOLT, GEORGE D.	Stagg, Cal.
O'BRIEN, P. A., Constancia Mines, Care Nicaragua Commercial & Logging Co.,	Princepulca, Nicaragua.
ORMSBEE, J. J.	Hayden, Ariz.
OVERPECK, A. C.	P. O. Box 416, Rapid City, S. D.
PAGE, WILLIAM KINGMAN, Care American Smelting & Refining Co.,	S. Chicago, Ill.
PATTERSON, R. C., JR.	55 W. 44th St., New York, N. Y.
PETERS, RICHARD	Care W. J. Rainey, 52 Vanderbilt Ave., New York, N. Y.
PINCHOT, GIFFORD	1214 Real Estate Trust Bldg., Philadelphia, Pa.
PREYMAN, F. R.	Oyster Bay, L. I., N. Y.
PROSSER, H. A.	120 Broadway, New York, N. Y.
PUTNAM, BENJAMIN R.	73 Plaza Drive, Berkeley, Cal.
RABLING, HAROLD	St. Joseph Lead Co., Bonne Terre, Mo.
RENO, CHARLES A.	Platora, Nev.

- RHODES, CLARENCE E. 475 N. Marengo Ave., Pasadena, Cal.
 RICKARD, B. N., Care American Smelting & Refining Co., Apartado 63,
 Chihuahua, Mexico.
 ROGERS, O. P. Box 114, Juneau, Alaska.
 ROUDABUSH, MARTIN A., Topographer, U. S. Geological Survey, Washington, D. C.
 SAULLES, C. A. H. DE 120 Broadway, New York, N. Y.
 SAXMAN, CHARLES W. 415 Depot St., Latrobe, Pa.
 SCHLACKS, CHARLES H., Care Hale & Kilburn Co., 18th and Lehigh Ave.,
 Philadelphia, Pa.
 SCHMIDT, WILLIAM C., JR., Care New York & Honduras Rosario Min. Co., San
 Juancito, Honduras, via New Orleans and Puerto Cortes.
 SCOTT, WALTER P., El Tigre Mining Co., S. A., Esqueda, Son., Mexico,
 via Douglas, Ariz.
 SEVERY, C. L. 322 N. Sixth St., Fort Smith, Ark.
 SHAW, J. S., Care Van Horn-Shaw onstruction Co.,
 1010 American National Bank Bldg., Fort Worth, Tex.
 SHELBY, W. W. P. O. Box 233, Bisbee, Ariz.
 SHIMER, W. R. 24 S. Center St., Bethlehem, Pa.
 SIBBALD, ALEXANDER 406 W. Granite St., Butte, Mont.
 SMITH, E. A. C., Care American Smelting & Refining Co., 120 Broadway,
 New York, N. Y.
 SMITH, F. G. D. Santo Domingo Silver Mining Co., Batopilas, Chih., Mexico.
 STEWART, JUDD, Care American Smelting & Refining Co., 120 Broadway,
 New York, N. Y.
 TARANTOUS, RICHARD, Asst. Supt. Chisos Mining Co., Terlingua, Tex.
 THOMPSON, A. P. Care Utah Apex Min. Co., Bingham Canyon, Utah.
 TOMBO, CARL Reading Terminal, Philadelphia, Pa.
 VAN ARSDALE, WILLIAM H. Hotel Windermere, Chicago, Ill.
 VAN DORP, G. H. 1115 Polk St., Topeka, Kan.
 VAN VALKENBURGH, R. D. Greene, Chenango Co., N. Y.
 VAUGHAN, FRANCIS E. 2231 Dana St., Berkeley, Cal.
 WEINBERG, ERNEST A. Care H. E. Verran, 583 Broadway, New York, N. Y.
 WEISENBURG, ANDREW Care Empresa Hanseatica, Barranquilla, Colombia.
 WELD, C. M. 60 Broadway, New York, N. Y.
 WENTWORTH, HENRY A., Cons. Engr. 60 India St., Boston, Mass.
 WESTERVELT, E. W. Instructed to hold all mail.
 WETHERED, ROY Chewelah, Wash.
 WHITE, L. L. Care Monarch Madonna Mine, Garfield, Colo.
 WILLIAMS, R. E. Ladies Mile, Remucca, Auckland, New Zealand.
 WILLIS, FRANK G. P. O. Box 119, Cripple Creek, Colo.
 WINSOR, F. H., JR. Mitchell, S. D.
 WISE, SIDNEY L., Min. Engr. 51 E. 58th St., New York, N. Y.
 WOODSON, E. F. Care J. R. Crowe, C. & M. Co., Weir, Kan.
 WORTH, JOHN G. Room 1416, 66 Broadway, New York, N. Y.
 ZEHNDER, C. H. Room 1648, 120 Broadway, New York, N. Y.
 ZIMMERSCHIED, K. W., Met. General Motors Co., Detroit, Mich.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

- | Name | Last address of Record, from which Mail has been Returned. |
|----------------------|--|
| BLOW, JOHN J. | 173 Rodney St., Brooklyn, N. Y. |
| BOYS, H. R. | Aurora Cons. Min. Co., Aurora, Nev. |
| CLAGHORN, JAMES L. | P. O. Box 1725, Bisbee, Ariz. |
| CRARY, CHARLES N. | Kimberly, Nev. |
| EATON, E. R. | Manatawny Bessemer Ore Co., Manatawny, Pa. |
| FINLEY, WALTER H. | Logmont, Ky. |
| GREY, GEORGE R. | 54 Ameshoff St., Johannesburg, Transvaal, South Africa. |
| HEBBARD, AUSTIN. | Box 98, Langlaate, Transvaal, South Africa. |
| HORCASITAS, A. S. | Care Tanawah Gold Min. Co., Sweet Water, Nev. |
| KENNEDY, GEORGE A. | 2741 Boulevard F, Denver, Colo. |
| KNOERTZER, H. | 56 Rue Balagny, Paris, France. |
| LANGLEY, SETH S. | 601 W. 190th St., New York, N. Y. |
| McKIM, JOHN W. | 532 Dooly Block, Salt Lake City, Utah. |
| MAINWARING, H. M. C. | Chillagoe, Ltd., Chillagoe, Queensland, Australia. |
| NOBS, FREDERICK W. | La Leonesa, Matagalpa, Nicaragua. |

RALPH, E. W.....	Care Boston Ely Min. Co, Kimberly, Nev.
REVELL, GEORGE E.....	P. O. Box 132, Nelson, B. C., Canada.
SALE, A. J.....	Giroux Cons. Mines Co., Kimberly, Nev.
STEEL, DONALD.....	202 Emerson St., Palo Alto, Cal.
SUNDERHAUF, A. E.....	Paramaribo, Dutch Guiana.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Apr. 10 to May 10, 1915:

Date of Election.	Name	Date of Decease.
1902	*McNULTY, S. G.....	March 13, 1915.
1906	*PARSONS, A. R.....	March 3, 1915.
1914	*SHEPARD, FREDERICK N.....	
1880	*SHERRERD, JOHN M.....	April 16, 1915.
1909	†SWANK, JAMES M.....	June 21, 1914.

* Member.

† Honorary Associate Member.

BIOGRAPHICAL NOTICE

John Maxwell Sherrerd was born Feb. 26, 1859, at Scranton, Pa. His father, Samuel Sherrerd, had risen as a lawyer to judicial position; but his two sons, Morris (now City Engineer of Newark, N. J.) and John, turned their attention to science and engineering. The latter entered the Pardee Scientific School of Lafayette College, at Easton, Pa., where he studied mining engineering under J. M. Silliman, and chemistry under that brilliant and thorough teacher, Thomas M. Drown. It was during this period that I made his acquaintance. I think he was a member of my class in economic geology; but I remember him better as one of "Drown's men"—a synonym for skillful and trustworthy chemists. But he was strong in mathematics also. In 1877, he won the mathematical prize of the Junior year, and in 1878, he graduated with honor as a mining engineer, after winning also as a Senior the distinction of membership in the Phi Beta Kappa. After two years, spent in the Columbia School of Mines, he became chemist for Ario Pardee at Secaucus, N. J., where his knowledge of that science was brought to bear, through the analysis of ores, fluxes, slags, and pig iron, upon the management of the iron blast furnace. Nowadays, the chemist runs the furnace, taking the place of the old-time, rule-of-thumb, tradition-worshiping "founder," and it is hard to realize that, 40 years ago, the continuous employment of a man like Sherrerd as a member of the staff of such an establishment was a revolutionary novelty. But the chemist, when he had once gained foothold, soon demonstrated his value in terms of commercial profit; and the chemical laboratory has become a recognized door to further promotion.

In 1882, Mr. Sherrerd took charge of the chemical department of the Troy Steel & Iron Co., at Troy, N. Y., a famous establishment, which was waging a long and losing fight, under crippling conditions as to fuel, freight, etc., against competitors more favorably situated. It is under such circumstances that science often delays surrender; and certainly in such a struggle the resources of the technical expert are more effectively exhibited than in the easy management of a highly profitable business. Mr. Sherrerd remained with the company until in 1895, after 13 years, it

went out of business. By this time, his reputation was well established; and he was at once engaged by the Taylor Iron & Steel Co., of High Bridge, N. J.—an establishment engaged in the manufacture of special steels, etc., adapted by peculiar chemical composition to particular uses. Here his scientific training was put to a new use. The company's products had to be brought to the attention of consumers under the direction of a representative who knew exactly what they would do or endure in practice; how they could be modified to meet special needs: in short, who could promise for them neither less nor more than could be fulfilled in practice. And so Mr. Sherrerd became General Sales Agent, developing in that position, which he retained for 18 years, a business capacity which, reinforced by indispensable technical knowledge, achieved notable success. Retiring from this work in 1913, he became connected with the Kennedy Stroh Corporation, of Pittsburgh, Pa., by which he was employed until he died.

In 1885, while residing at Troy, Mr. Sherrerd took two important steps. He became a member of the A. I. M. E., and he married Miss Hawley of Troy. I may be pardoned for coupling these events; for, in fact, they are inseparably connected in the memory of his friends, who have enjoyed for many years at the meetings and on the excursions of the Institute, the welcome presence and genial fellowship of Mr. and Mrs. Sherrerd.

He was a member of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the Lake Superior Mining Institute, the National Geographic Society, the American Society for Testing Materials, and the Engineers' Club of New York.

Mr. Sherrerd died Apr. 16, 1915, at his home in Easton, Pa., after an illness of only two weeks, the complicated and fatal character of which had not been foreseen. His wife and five children survive to mourn him, in company with a multitude of friends, comprising all who knew him.—R. W. R.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*, JOHN H. JANEWAY, *Vice-Chairman*.
 F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
 P. A. MOSMAN, *Treasurer*.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*, ALFRED C. LANE, *Vice-Chairman*.
 HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
 ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.

FRANK A. ROSS, *Chairman*, RUSH J. WHITE, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.

I. F. LAUCKS, *Chairman*, J. F. MENZIES, *Vice-Chairman*.
 GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
 W. C. BUTLER, H. L. MANLEY.

Southern California

THEODORE B. COMSTOCK, *Chairman*, SEELEY W. MUDD, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
 A. B. CARPENTER, C. COLCOCK JONES.

Colorado

CHARLES A. CHASE, *Chairman*, S. A. IONIDES, *Vice-Chairman*.
 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*, JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.

G. HOWELL CLEVINGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
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Modern Gas-Power Blower Stations

BY ARTHUR WEST, SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1915)

It is the purpose of this paper to describe briefly some recent large power stations for blast furnaces, where the blast is exclusively supplied by gas engines using furnace gas. The stations are given in the chronological order in which they were designed, in order to indicate the various improvements suggested by experience.

Fig. 1 shows the blowing-engine station of the Bethlehem Steel Co. at Bethlehem. Although the picture shows only five blowing engines, the house now contains 11 engines. In the diagrammatic plan the arrow shows the position and direction of camera. This station handles five modern blast furnaces of 450 to 500 tons capacity each. In this, as in all the other stations described in this paper, each furnace is blown by two single tandem gas engines. Ten engines are thus required for five furnaces, leaving one single tandem spare engine. This spare capacity has been shown by several years' experience at Bethlehem to be adequate to insure continuous operation. The engines in the Bethlehem station are right hand and left hand, arranged in pairs. The outboard bearings of each pair are close together so that there is not room for passageway between them. On the right of the picture are shown the inclined drums communicating with the cold-blast lines outside of building, so that any engine may be operated on any furnace. The illustration shows four cold-blast gate valves with pipes passing through wall, but since the photograph was taken the drums have been extended to take care of a fifth valve for a fifth furnace.

At the extreme back of the picture can be seen the tubs of three twin vertical-horizontal Southwark steam blowing engines, which, although of modern design and as good as new, are not operated because of the greatly reduced blowing cost shown by the gas blowing engines.

The next station built is shown in Fig. 2. This is the plant of the Minnesota Steel Co. at Duluth, Minn., and is at present constructed for five engines, to handle two furnaces, with one spare engine. The engines are of practically the same size as those at Bethlehem. Instead of being arranged in right-hand and left-hand pairs, as at Bethlehem, all the engines are of the same hand; this arrangement has some ad-

vantage over the Bethlehem plan, in that it allows passageway between adjacent engines instead of only between alternate engines. The Minnesota plan, however, requires a longer house for the same number of engines than does the Bethlehem plan. There is no special operating advantage in having the engines all of one hand, because these engines are designed so that all parts subject to renewal are independent of the hand of the engine.

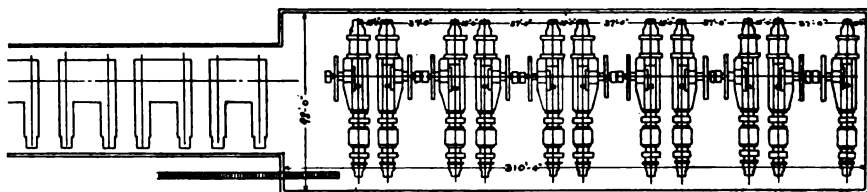
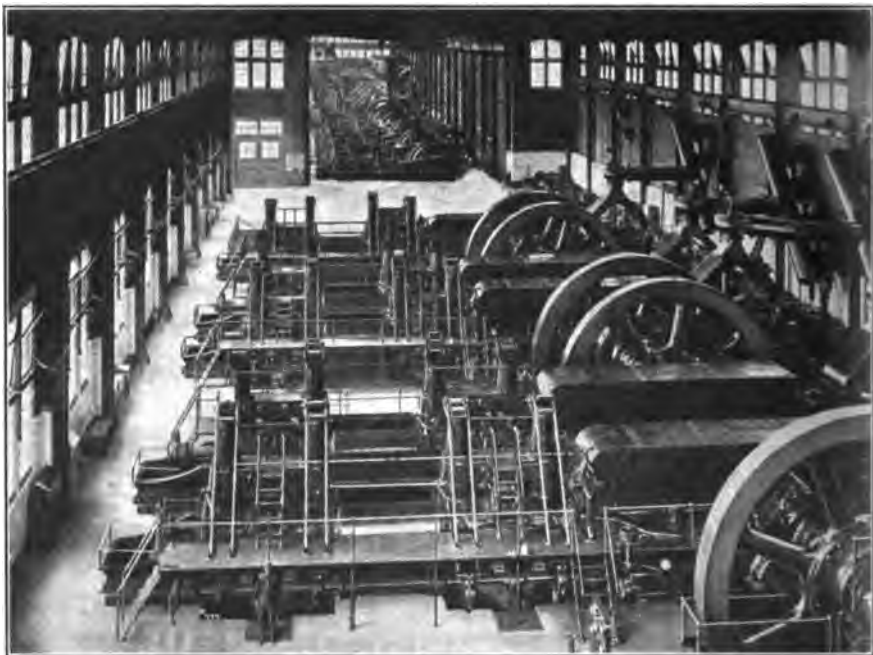


FIG. 1.—BLOWING-ENGINE STATION OF BETHLEHEM STEEL CO., BETHLEHEM, PA.

This station has been completed some months, but has never been run, as the new steel plant is not yet in operation. No steam blowers have been provided to blow in the new furnaces; the blowing engines will be operated on gas from the open-hearth producers until blast-furnace gas becomes available.

Figs. 3 and 4 illustrate the blower station of the Maryland Steel Co. at Sparrows Point, Md. These engines are the same size as those of

the Minnesota Steel Co. As the photographs and plan view show, the engines are arranged in right-hand and left-hand pairs, as at Bethlehem. In this case, however, the engines are set diagonally in the house. It will be seen that with this construction the outward crank-shaft bear-

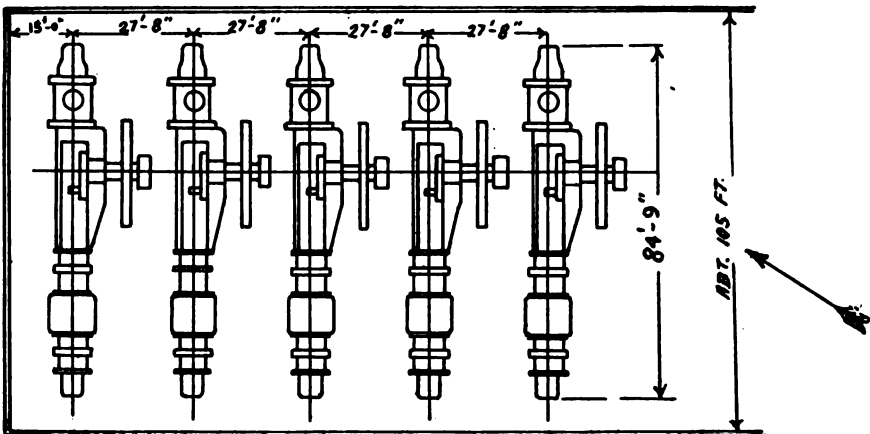


FIG. 2.—MINNESOTA STEEL CO. BLOWER STATION, DULUTH, MINN.

ings overlap. At the same time, as the picture shows, these bearings are far enough apart to allow ample passageway between them. Although the engines are the same size as at Bethlehem and Minnesota, the diagonal arrangement at Maryland gives more room around the

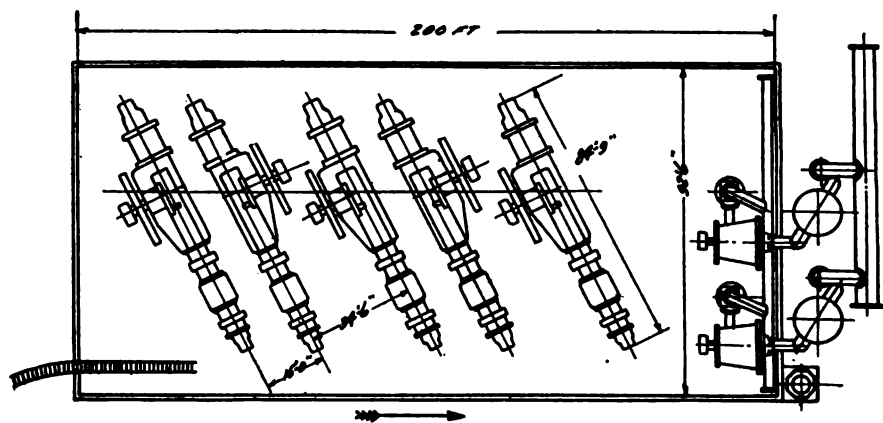
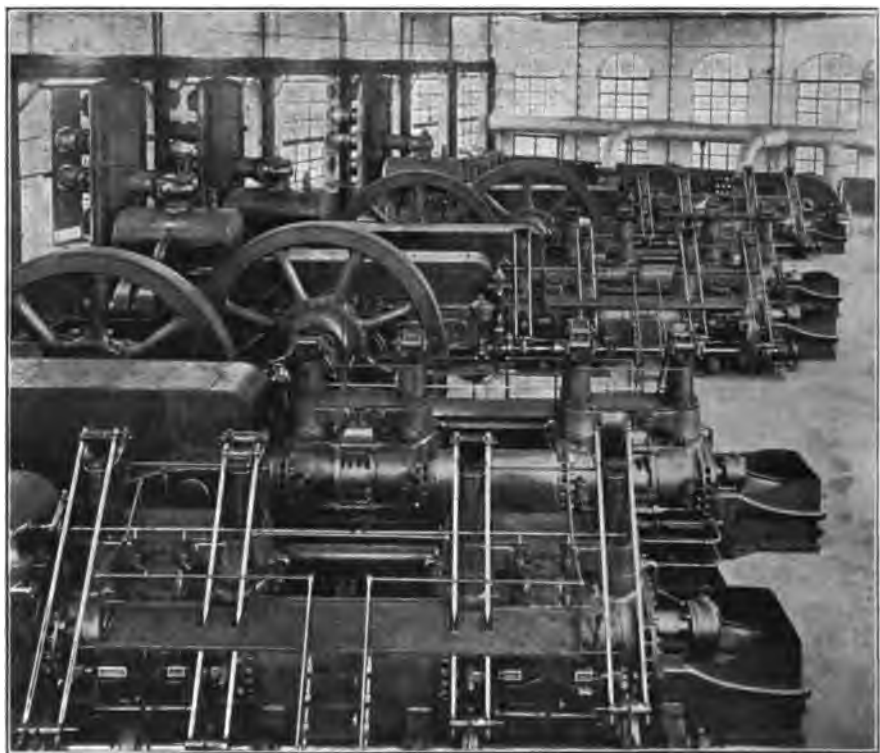


FIG. 3.—MARYLAND STEEL CO. BLOWER STATION, SPARROWS POINT, MD.' LOOKING TOWARD THEISENS.

engines, with a power house 20 ft. narrower. The length of the house is not increased, both because the outward bearings lap by each other, and because room remains in the triangular space at the end of the station for a railroad car under the traveling crane. With the engines set squarely across the house, at least one extra panel must be provided to allow the entrance of such a car. The power house at Maryland being 20 ft. narrower than at Bethlehem and Minnesota is, of course, less expensive, both because the area of the power station is

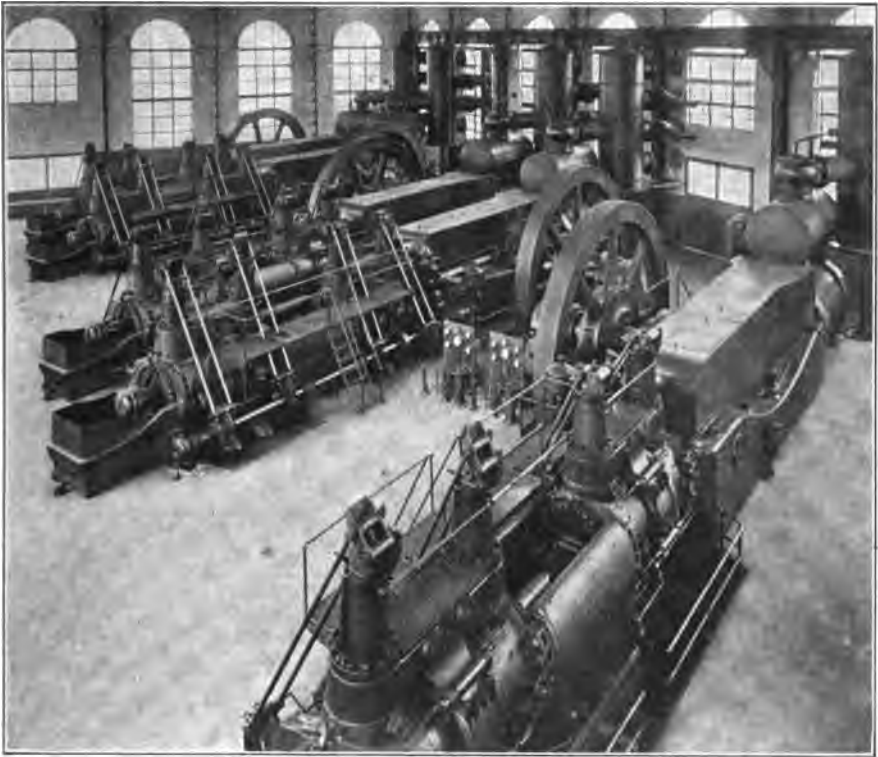


FIG. 4.—MARYLAND STEEL CO. BLOWER STATION. LOOKING AWAY FROM THEISENS.

less and because the cost of the building per square foot is less, the cost of each roof truss and of the traveling crane being reduced because of the lesser span. As shown in the illustration, the gauge boards and operating apparatus are brought together for a pair of blowing engines. The cold-blast valves and snorting valve on each blower are operated by compressed air, controlled from in front of the gauge board. Without moving from the front of the gauge board, one man can single handed put two engines on a furnace very quickly. The operations are:

1. By means of compressed air, open cold-blast valve to proper furnace, thus letting the blast pressure directly on to the Dyblie automatic check valve shown on top of tub.
2. By same means, open air snorting valve shown at side of tub.
3. Open jacket-water valve. Water will be seen discharging into funnel shown at rear of tail slide of engine.
4. Throw in igniter switch.
5. Open gas throttle.
6. Open valve supplying compressed air for starting gas engine. The engine will immediately pick up speed and run under the control of the governor, the tub discharging its air through the snorter out of doors, the Dyblie valve remaining seated under the blast pressure.
7. By means of compressed air, close snorter valve. The Dyblie valve automatically opens wide and the engine is blowing the furnace. The engine being all the time under the control of the governor, its speed does not vary when it is put on to the furnace. It being quite unnecessary to warm up as in the case of a steam engine, such a gas engine can be put on a furnace very rapidly, always inside of 1 min. I have seen it done in 35 sec., the engine being absolutely cold. It is unnecessary to adjust any oil feeds on the engine as all oil supply automatically stops and starts with the engine.

This station contains five engines for two furnaces, but is designed to be lengthened to allow the installation of four more engines, thus providing nine engines for four furnaces. As shown, the blast drums are provided with four openings for four furnaces.

Fig. 5 shows a nearer view of the speed-governing apparatus. H. A. Brassert, in his paper before the American Iron and Steel Institute on Modern American Blast Furnace Practice, stated that "*uniformity of conditions is the keynote of successful furnace operation.*" He also calls attention to the fact that of all the elements introduced into a furnace in a given time, air is the greatest, both by volume and by weight. It is evident, therefore, that the air should be measured into the furnace as carefully and as uniformly as are the ore, coke, and limestone. Now, a good modern blowing tub with automatic valves is, outside a holder, the best known means for measuring air. With properly designed air valves, leakage is infrequent. If such leakage exists, it can be detected by ear immediately and can be easily remedied. Since the production capacity of the blast furnace, per unit of time, must be held constant, even though the blast pressure varies over wide limits, it follows that the weight of air delivered to the furnace per minute must also remain constant no matter what the variation in blast pressure. For this reason all the blowing engines described in this paper have air

tubs designed with the utmost care. In addition, the engines are equipped with speed governors as sensitive as those employed on the same gas engines operating on 60-cycle current in parallel. In Fig. 5 is shown such a governor equipped with means for holding the engine at any desired speed, irrespective both of fluctuations of the blast pressure and of heat value of the furnace gas. Without such care in the governing apparatus, a change in either or both of these conditions will cause a change of speed and a consequent variation in the weight of air as compared to the weight of coke charged to the furnace. This is one of the foes to that uniformity desired by all successful furnace managers.



FIG. 5.—MARYLAND STEEL CO. BLOWER STATION. NEAR VIEW OF GOVERNOR.

To the left of the governor is shown a sensitive recording speed chart. This shows just how much air per minute was being charged to the furnace at any given time. Incidentally, it also gives the exact time and duration of furnace checks.

In all the furnace gas engine power plants of which the writer has knowledge, either in this country or in Europe, it has been the practice to place the apparatus for cleaning the engine gas far from the engine house. When, however, the Maryland Steel station was being laid out, the writer recommended the installation of the Theisen washers (Fig. 6)

directly in the engine room. Since the economical operation of the gas engines is dependent upon the cleanliness of the gas, it seems best to charge the chief engineer of the power station directly with the control of and responsibility for the engine gas-cleaning apparatus. This arrangement has proved very successful at the Maryland plant. It only required the addition of one bay to the length of the station and saved nearly the whole first cost of the Theisen house, with its traveling crane, etc. It also concentrates responsibility and saves operating labor, besides being agreeable to the safety department.



FIG. 6.—THEISEN WASHERS AT THE MARYLAND STEEL CO. BLOWER STATION.

Each of the Theisens shown cleans gas enough for five gas engines, or more than enough to blow two furnaces. With the Maryland station of its present size, one Theisen is operated, leaving the other as a spare. When the station is enlarged to care for four furnaces, a third Theisen will be installed in the vacant space to the left. Two Theisens will then be regularly operated, leaving the third as a spare.

Fig. 7 shows the outside appearance of the Maryland blower station. The pipe on the right brings raw gas to the station; the gas having been cleaned only so much as is required for the stoves and boilers. Two towers can be seen just outside the station at the right, each tower being in series with each Theisen. The tower is empty and is provided with water spray, up through which the gas passes on its way to the Theisen.

The tower does some cleaning, but should principally be regarded as a cooler.

The Theisens clean the gas very effectively and are not at all extravagant in their consumption of power, which is about 2 per cent. of the power of the engines they supply. A new type of disintegrator is in use in Europe which is said to consume very much less power and water than the present Theisen apparatus.

At the right of the picture is seen the exhaust stack, which connects to an exhaust tunnel, incorporated in the concrete foundation of the building. A certain amount of salt water is discharged into this exhaust tunnel to keep the temperature below 212° . Such an exhaust tunnel is a perfect silencer and is in successful use at Gary, at Bethlehem, and at Sparrows Point.



FIG. 7.—OUTSIDE VIEW OF MARYLAND STEEL CO. BLOWER STATION.

The station of the Maryland Steel Co. is also provided with a cooling pond, located just to the right of the exhaust stack and in line with the power station. This was provided because only brackish water was available, which it was not considered desirable to use in the gas engines. The cooling pond has no towers in connection with it and works very satisfactorily, requiring the operation of only one centrifugal pump in addition to the apparatus that would otherwise be required. In cases where water is very bad or very scarce, such a gas station with a cooling pond will furnish power much more cheaply and simply than can any steam plant. In fact, a careful examination of the Maryland plant will convince an experienced blast-furnace manager that such a gas-power station is not only much more economical than a steam station, but is much simpler; the boiler plant, condensing apparatus, etc., being entirely eliminated.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Fire-Fighting Methods at the Mountain View Mine, Butte, Mont.

BY C. L. BERRIEN,* BUTTE, MONT.

(San Francisco Meeting, September, 1915)

MANY fires have occurred in the mines of Butte in recent years, and while all have been of a serious nature, simply because they were mine fires, six of them have been especially dangerous in respect to loss of life and property, and the expense of extinguishing or controlling them. It is not my intention to discuss the relative merits of methods used in fighting these different fires because the conditions under which they occurred varied in each case, and it was not always advisable to expend the large sum of money necessary to actually extinguish them if they could be bulkheaded off from the rest of the mine. It is needless to say that everything possible has been done to check all fires at the beginning, but without success in some cases; and it has then become necessary to bulkhead all connections to them, in which state several now exist.

The mine fires of Butte, or fires in any mine, may be divided into two classes, namely: Those occurring in accessible workings, and those occurring in inaccessible workings; because these two conditions govern to the greatest extent the methods of fighting fires. Accessibility and inaccessibility in the case of mine fires concern not only new open workings and old closed workings, respectively, but also the condition of the ventilation in and about those places. When a fire is accessible at the start it should not be difficult to confine it to a small area and extinguish it in a relatively short time, but when it is impossible to actually reach a fire in old filled, abandoned workings, or there is no opening by which the gases may be conducted away from the fire zone, then the time consumed in changing those conditions permits the fire to increase to such an extent that it is a long, hard fight to overcome it.

These different conditions have been met in Butte, and of the six fires of serious proportions the Steward mine fire and the High Ore-Modoc mine fire were in new accessible workings and were extinguished in a month or so. The Anaconda mine fire, the Leonard-Minnie Healy mine fire, the West Colusa mine fire, and the Mountain View mine fire, were all inaccessible. The Anaconda mine fire started about 25 years ago and though safely bulkheaded is still active. The Leonard-Minnie Healy

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fire is about eight years old, bulkheaded and active. The West Colusa fire began about two years ago, is now bulkheaded, but is still alive. The Mountain View fire originated in June, 1913, and at the end of December, 1913, to all appearances was extinguished.

I had been at the Mountain View mine for three years as assistant foreman, and previous to that was in the geological department about six years. In that time my work took me to many different mines of the company, and owing to the existence of other mine fires the smelling sense in regard to the resultant gases had become well educated.

Aside from that, I had always noticed that each mine, or different places in the same mine, had a distinctive, though possibly faint, difference in odor. At any rate, a man knows or should know the odors of the mine in which he works and investigate the slightest change in them. Of course any one can notice decided changes, but it is the first faint odors to which I refer, and the reason I mention this as being especially important is because a month or so may be gained or lost in starting work toward the source of danger when the fire is not an open one. Furthermore, the fault of not recognizing these things immediately, or of not thoroughly investigating the slightest suspicion, may lead to loss of life, or something less serious, as in the case of the West Colusa fire, when gases from there came into the Mountain View mine on night shift and bosses failed to report it because they were not positive. As a result 30 men on the day shift were overcome when they went to their working places.

I say the Mountain View fire originated in June, but by that statement is meant that at that time we actually knew a fire existed. It is a fact that for some weeks previous to that date we had our suspicions that something foreign was in the air of the mine which could not be traced to any definite location. On two successive change days when only repair men were working and the air of the mine was less disturbed in places, because no air drills or cages were running in the mine, the writer detected a faint odor of gas which could not be traced then and which disappeared entirely when the mine was working. On June 18, 1913, several shift bosses of the day shift and the mine foreman had detected the smell of wood smoke, but not being positive asked the writer, then on night shift, to investigate.

They thought the trouble was near 550 crosscut, the location of which is shown on Plate 8, so I went there first and at the top of B-614 raise could smell wood smoke. Going down the raise 10 ft. and west 50 ft., along the top of the stope filling, a haze of wood smoke was evident. This stope had been worked out and filled all but 10 ft. under the 500-level sill six months or so before. West of the cross mark was a solid gob of waste and square-set timbers 12 ft. wide, six years old, and the smoke was coming from below and west of that point. The smoke was not thick enough at that time to harm a person, but was visible and sta-

tionary, there being no movement to it on account of everything being filled below and west.

We had a fire, and the first move was for the safety of the men, and the next for water. The ventilation of this part of the mine was by air traveling up, and in its course was passing 30 men working in east-end stopes from the 500 to the surface. The shift boss on that run was sent to bring those men to the 500 while fire hose was brought from the surface, attached to the water pipe in the main air shaft and run to the stope at the cross. It took about 20 min. to do this and inform officials of conditions. There was no fire, smoke, or gas on the 600 in drifts directly under the cross nor were there any other workings open on the 500 by which we could get nearer the cross aside from A-557 drift and 550 crosscut. All workings south of A-557 and west of 550 as far as Fan 3 were caved or filled. (See Plate 8. Solid lines on level maps show workings before the fire. Dotted lines show work done after fire started.)

Feeling positive that the fire was west of the cross, the men were removed from the stopes above and put to work immediately to clean out A-581 and 586 in order to get water to the stopes under 515 drift. At the same time similar work on the 400 level was started at A-445 and 412 off 449, and at A-423 and A-466 off A-462, to get water into stopes under 416, the old hanging-wall drift. (See Plate 5.)

The first work the following day was to change the entire ventilation on the east end of the mine so that we could draw the gases out of the mine and continue to operate. We had the two-compartment Sullivan air raise, each compartment being 5 ft. square in the clear, on the east end of the mine, which at that time was downcast.

At the collar of this raise was installed a No. 15 Sirocco fan changing it to a strong upcast, which later enabled us to work toward the fire from the main part of the mine. For eight days we worked along these lines, getting water as close to the source of the fire as possible, but instead of the smoke decreasing it gradually increased until we could not go below the sill of 550 crosscut. During that time, however, we had put in a 2-in. pipe line for water on the 400 level from the main air shaft to A-476 crosscut by way of A-479, A-409, 495, A-452, 449, A-462, and 441, with tees at each crosscut leading to the south. On the 500 we had put in a 2-in. line from the main air shaft to the junction of 550 crosscut and A-576 drift by way of 591, 594, A-583, A-557, and 550, with tees at each crosscut. These pipe lines were hung to the timbers in the drifts. We had also put in a 2-in. line on the 600 from the main air shaft to a point under the fire on the 500 by way of A-622, A-645, and B-642, where we kept men all the time to watch for fire from above. These lines were all connected to a 2-in. water line in the shaft coming from the city water line on the surface. The pipe was made to stand a pressure of 440 lb. The 600 level is 887 ft. below the collar.

All the time we had Draeger helmets of the 1904-09 and 1910-11 types ready for use, and a man on each shift to keep them repaired. We had all had practical experience in using them.

On the ninth day of the fire the writer took charge of the mine. Conditions were growing worse each day and we realized the necessity of starting in systematically to protect the rest of the mine and getting water through all the stopes from the top to the 600. It was evident at the time that the fire would spread east and west and to the levels above and below before we could cut it off.

We decided to run lateral drifts east along the old south drifts on the 200, 300, 400, and 500, and directly over the stopes below, keeping the old timbers in sight on the left of the lateral and solid ground on the right. We further planned to put water into the old workings as we advanced and at certain intervals to run crosscuts south where we could set up diamond drills to drill holes to the stopes below. We knew we must get water into the stopes elsewhere than directly from the levels because some of the stopes were so wide and flat that the water from levels would only cover the foot wall. To carry out these plans we needed air pressure to force the gas and smoke ahead of us, and lots of water.

A new 4-in. pipe for water was put in the main air shaft and with the 2-in. column already there we had enough water to supply five 2-in. lines on the levels at full capacity. One line was run on the 200 level to the Sullivan upcast, one line on the 300, a new line on the 400, where we already had one, a new one on the 500, where we already had one, and with the one on the 600 gave us seven 2-in. lines, which were brought along as we opened new country in the fire zone. We also installed a No. 4 Sirocco fan on the 400 level at the junction of A-409 and 495, one at position marked fan No. 1 on the 500 level, which later was moved to position of fan No. 2, and one at position marked fan No. 3 on the 500 level. The air was conducted from these fans by galvanized-iron pipe 10 in. in diameter, where it was to serve one place, and by 18-in. pipe branched to 10-in. where the fan was to serve more than one place.

To maintain the air pressure in the working places we always built two bulkheads 15 ft. apart so that one could always be closed while men were going and coming. These bulkheads consisted of a tunnel set of 10 by 10 in. posts 7 ft. long with only a 3-in. or 5-in. cap; the space between the set and the ground was concreted.

The galvanized pipe, air pipe, water pipes, an extra pipe for emergency, and electric-light wire pipe rested on the cap and were concreted in place. The doors were of wood and opened inward so they would always close with the pressure. A flap of canvas was nailed on the bottom of each door to prevent the air from escaping. Aside from these working doors a solid concrete bulkhead with sheet-iron doors opening outward was built in each drift for safety in case the fire drove us back.

Speed and safety were the principal objects in the work and the less ground we had to move the faster we went.

Before describing the details of work performed it would be well to mention conditions in general from the tenth to the twentieth day. The mine was producing at that time 1,600 tons a day and throughout the fire period never missed a shift. We were then running 200 gal. of water a minute on to the fire, which after the first month was increased to 670 gal. per minute. This rate was kept up for five months and the water carried about 1 per cent. copper when it reached the pumps.

Bulkheads, dams, and water boxes had to be put in on levels below the 600 to the 1,400 to handle the water coming from the fire zone, and this with the repair work gave us a great deal of trouble. On the eleventh day, fire was discovered at the cross on the 600 level (see Plate 10) in the timbers over the drift and along the back to A-696 drift, making 20 ft. of fire. The ground above was stoped and filled over A-693, A-696, 623, A-635, and A-655, and gave an excellent opportunity for fire to descend. Water was then coming down to the 600 through some of these workings. The only open drifts under the fire on the 600 were B-645, B-642, A-696, A-690, and B-616.

Looking at Plate 11, which shows the workings one floor under the 600 level, one can see the possibilities of a fire to pass the 600 and go on down through the mine. For five days we had an interesting time keeping the fire above the level and it was necessary finally to have two hose lines on the work there with six men on each 6-hr. shift. Aside from that it kept four men busy all through the fire period keeping the drifts open, which were very heavy on account of water coming from above. For several months, until we ran a new lateral drift south of the old workings, a man going in through B-642 to the east could come out only the way he went in, and with the smoke and fire along the back the men on the 600 had a far from pleasant time. Fearing that the fire would drive the hose men out we laid lines of perforated 2-in. pipe, wrapped in tarred canvas, along the bottom of all the open drifts, our intention being to run water through them should access to that country be cut off.

Because the fire and the work on it were assuming such proportions, it was decided to separate them from the regular mine work; and mine fire bosses, three on each 8-hr. shift, were appointed. The assistant foreman and regular mine bosses took care of the ordinary operations in the mine.

On the 500 we had reached 523 drift in A-581 crosscut and had installed diamond drill No. 1, with which we had drilled several holes, from all of which the core barrel came out almost red hot. (See Plate 9.) Gas and sulphur smoke were plentiful and we were working under pressure from fan No. 1. These fans with the air discharging inside our double

doors kept back the gas and smoke in nearly all cases, but when they failed we used the compressed-air system in conjunction with them.

Access to 550 crosscut was possible with Draeger helmets only. Water from the 400 was hindering the work considerably. On the 400 level we had run B-430, A-445, 412, A-423, and A-466 crosscuts in over the stopes and were putting down water at those points. In these places the men wore hoods on the back of which a small hose was connected for the use of compressed air. On the inside of the hood, over the connection, a small square of canvas was attached by its four corners to prevent the air from blowing directly on the head and to send the air around the face. These hoods were used in much of the work where a Draeger helmet was not needed. Carbon monoxide and dioxide were present on the upper levels from the beginning of the fire.

On the 300, during this first 20-day period, the gases appeared only on the east end of the mine on their way to the surface through the Sullivan raise.

A dam was built near the junction of 319 and B-314 over which water was run continuously. Nearly all workings east of that point were inaccessible because of being broken down or full of gas.

On the 200 up to that time we could go into the Sullivan raise, where for some weeks we judged conditions below by the smoke and gas passing through it. We had water piped to it but did not turn any in until some time later, as we feared we would weaken the raise and lose our upcast.

Along with our general plan of hanging-wall laterals and diamond-drill holes, as a means of quenching the fire entirely, we decided to build a fireproof wall west of the fire zone from the 200 to the 600 and a fireproof floor on the 600 under the fire to save the mine west of and below those boundaries should we fail to control the fire. This work was started by raises from the 300, 400, 500, and 600 levels so arranged that they made an almost continuous raise from the 600 level up. The details of this work will be explained later.

Throughout the fire period all of the work accomplished was carried on with some interruptions and many difficulties due to the increasing area of the fire and gases.

An account of all operations on different levels after the first 20 days will now be given.

On the 200 level (see Plate 1) for the first month we had access to the Sullivan raise, but about that time the ground around the top of A-322 raise caved, due to timbers burning out below, and left a hole 20 ft. wide over which we could not pass. The hole was bridged with three 15-ft., 10 in. by 10 in. stringers tailed one upon the other and the hole filled with waste and concrete. All the work was done by men working in Draeger helmets and was the hardest work of that kind we had.

The vein in 266 drift had been worked out from the 300 to the oxidized zone three floors above the 200, and west to within 10 ft. of 259 under the 200. Above the 200 the stopes, two floors high, ran west of 259 and over 296. At the junction of 259 and 266 we built a concrete bulkhead 35 ft. long, 20 ft. high and 4 ft. thick. This work was done in Draeger helmets, and on the top 5 ft. the men had to wear heavy gloves on account of the heat; it took two months to build the wall but we had the drift free from gas and smoke. By that time we could only get to within 60 ft. of the Sullivan raise because the drift had caved for that distance.

Fire had now found its way up to the 200 in the Sullivan raise, and it had become necessary to stop the fan and turn water down the raise from the surface or lose the entire raise. The fan was not run after that, as we were able to keep the gases back with our underground fans. We had cut off all unnecessary air from the fire zone below and on the 200 some of the downcast air from the main air shaft was allowed to travel east and up the Sullivan, thus holding back the gases on the fire below. As a result, we maintained a condition of rest for the air in the fire zone, which in my opinion was one of the greatest aids in the work. We could open the doors on the 200 level 6 in. more than required to maintain these conditions and force the gas back against the fans on the 400 and 500 levels. The miners on the 200 often demonstrated this for us by their carelessness in passing in and out.

A lateral drift, A-254, was run 20 to 30 ft. south of 266 and from it crosscuts were driven every 15 ft. to within 5 ft. of the old workings. Into the face of each crosscut two holes were drilled to pierce the old workings, and through each hole 1-in. pipe for water was run. This gave us 14 crosscuts and 28 water pipes, from which we soaked the stopes below. (See Plate 1.) Dotted lines show workings run for this object. Crosscut A-275 was run to discharge water into the stope below 259 drift.

Just east of the main air shaft on the 200 may be seen the top of fire-protection raise B-385, which was run up through the old gobs. From there A-268 crosscut was run south, and at the end a new air raise was run 450 ft. to the surface to be used for ventilation after the fire.

On the 300 two lateral drifts, B-390 and B-381, were run from B-353 and A-380 respectively to pass over the stopes below. (See Plate 2.) Dotted lines indicate the new work on fire.

We followed the old filled drifts where it was possible and ran water into them as we advanced. When A-368 was reached we found water coming down from the 200 and decided it would be unnecessary work to advance farther. Near the south end of B-383 crosscut we put up fire-protection raise B-385 through old gob and farther north in the same crosscut, but not showing on the map, we ran another raise to the 200 through old gob.

The work on the 400 level was more extensive. (See Plate 5 for loca-

tion of fire drifts, diamond drills, and fan.) The dotted lines show work done to reach the fire zone.

As stated, a Sirocco fan had been set up as shown with galvanized pipe leading into B-419 and around to A-445. Crosscuts B-430, A-445, 412, A-423, and A-466 had been run to the old stopes. We ran B-419 in about 100 ft., crossed the old drift, 416, and from there continued east over the old stopes below until opposite A-466 crosscut. In one month, 315 ft. were run and timbered by four 6-hr. shifts with five men on a shift. Two men drilled with a cross bar, one shoveled, and two ran car. B-422, B-425, B-426, B-421, and B-423 were run at the same time the drift was advancing.

Water was put down in all places as fast as it would sink away. Our belief was that the fire would beat us to the 400, and so it proved, for at the end of B-419 we ran into a mass of fire. A-462 at that time was filled with dense white smoke in which investigation work with Draeger helmets was very dangerous.

All of these workings were difficult to maintain on account of water from above.

Diamond drills were set up as indicated on Plate 5. The courses and dips of all these holes were set with a Brunton compass by the writer. The position of the drills and the starting points of the extreme holes of each series of holes were located before the drills were set up. Having these accurately located the intermediate holes were easily placed so that the country below would receive water at every 10 ft. From these three setups the holes shown on Plate 6 and Plate 7 were drilled. From diamond drill No. 3 the upper holes on Plate 3 and Plate 4 were drilled; also from drill No. 3, a series of holes was drilled to the 500 sill along 515 drift.

As soon as each hole was finished it was piped and water turned in immediately. One of the hardest things to do was the piping of some of these holes because of the gases which came up through them.

It is impossible for one to realize what a busy place B-419 drift was during these times and especially during the installation of apparatus, etc. All the pipes had to be wrapped with tarred canvas to protect them from the strong copper water; the rails would become eaten out; the side lagging break in, and other innumerable things happen to delay the work.

In between times, we had trips to make in Draeger helmets to different places, to note conditions, etc., through drifts a foot or two deep with water and other obstructions, or filled with sulphur smoke.

We ran B-427 just east of drill No. 3, from which place B-428 and A-488 crosscuts were driven. All the timber had been burned out there and the ground was so hot that it took 15 min. to cool the drill holes for safety in blasting. In B-428 we had the only accident during the fire.

One of the upper holes from diamond drill No. 3 had holed to the stope above and the water from it, turning to steam, caused an explosion that blew hot dust down into the crosscuts. Two men working in B-428 were very badly burned.

Near the fan, fire-protection raises B-431 and B-432 were run up through the old gobs.

On the 500, we had reached 515 drift in A-581 and had drilled six holes from diamond drill No. 1 as shown on Plate 9. The fire spread so fast that we lost the drill and all the rods and were driven out completely. Fan No. 1 was moved back to fan No. 2, where we built large concrete bulkheads with iron doors, beyond which we could not go except with Draeger helmets.

Work on the 500 was then confined to running 515, B-536, B-538, and B-540 as shown by dotted lines on Plate 8.

Diamond drill No. 3 was set up and the holes as shown on Plate 9 were drilled. The stope was full of fire and very difficult to extinguish. Many times we were driven out of 515 by the steam and smoke caused by water running from the drill holes on the fire.

We were very much concerned at this point because the fire was working west between the 500 and 600. Drift 515 was being extended all this time, the face of which was in burning timber a long way before our water from above hit it. In the last 100 ft. of 515 we ran into many of our water pipes which we had run down in drill holes from the 400.

Diamond drill No. 2 was set up in B-548 crosscut, which was run especially for the purpose from a south drift, it not being thought safe to place it anywhere along 515. The dots on Plate 9 indicate the holes drilled. We had intended to run a row of holes to the fifth floor of these stopes but this was unnecessary.

During the worst period on the 500 we decided to try steam, so a 3-in. line was put down the shaft and branched to two 2-in. lines; one of which discharged east of the bulkhead by No. 2 fan and one inside the bulkhead in B-535 near No. 1 diamond drill. From observations, made in Draeger helmets, we decided the steam did not help much and so gave it up.

At fan No. 3 and 50 ft. north of it, we put up two fire-protection raises through the gobs to the 400.

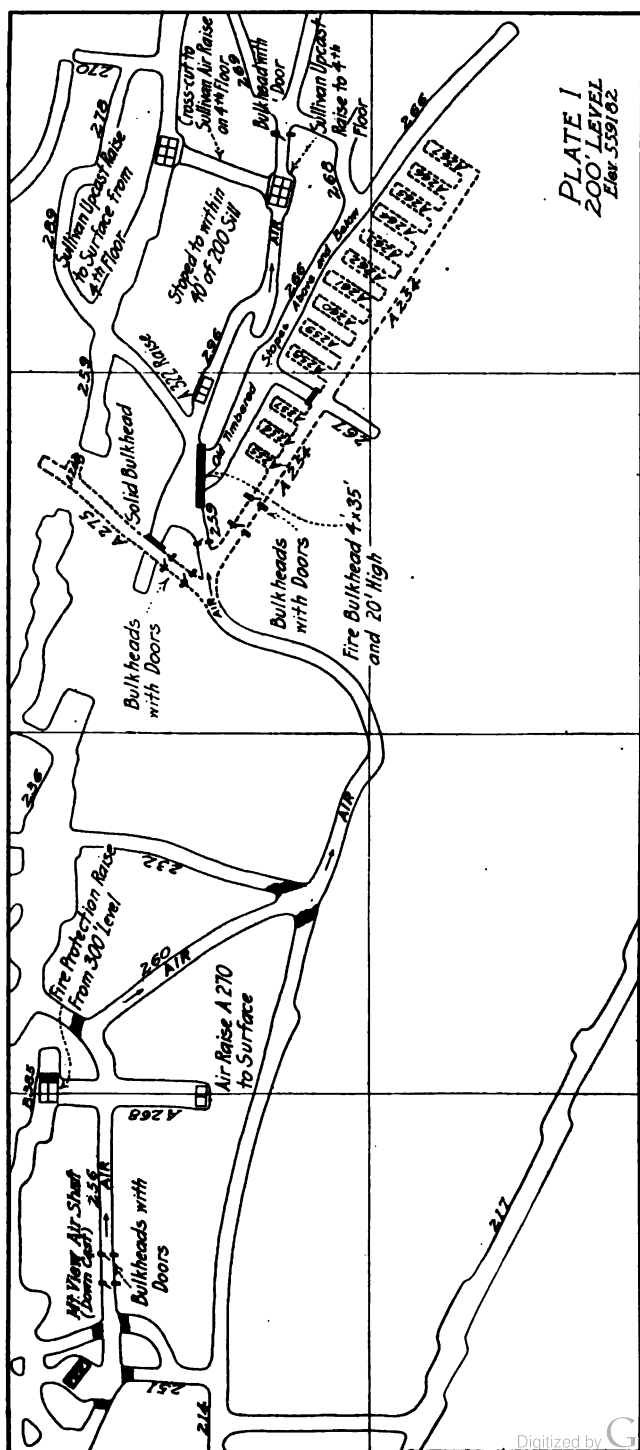
The work we had planned for the 600 sill seemed endless when we started, as water was coming down in such volume. Plate 11 shows the stopes and raises which reached the 600 from the 700. The dotted lines show the drifts which were run during the fire to reach these places. The plan, as stated before, was to make a fireproof floor 8 ft. thick by removing all timbers in the stopes, drifts, and raises, on the 600, and filling in with waste and concrete.

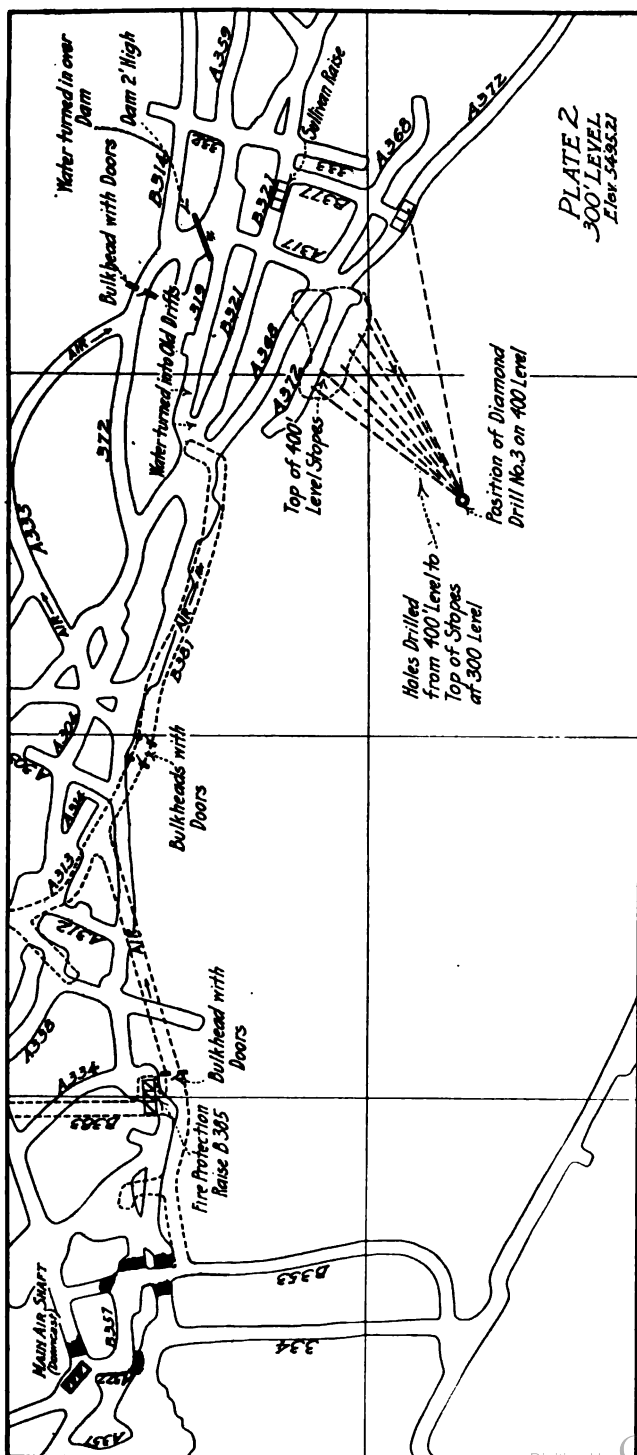
The first step was to run these new drifts through the old broken down

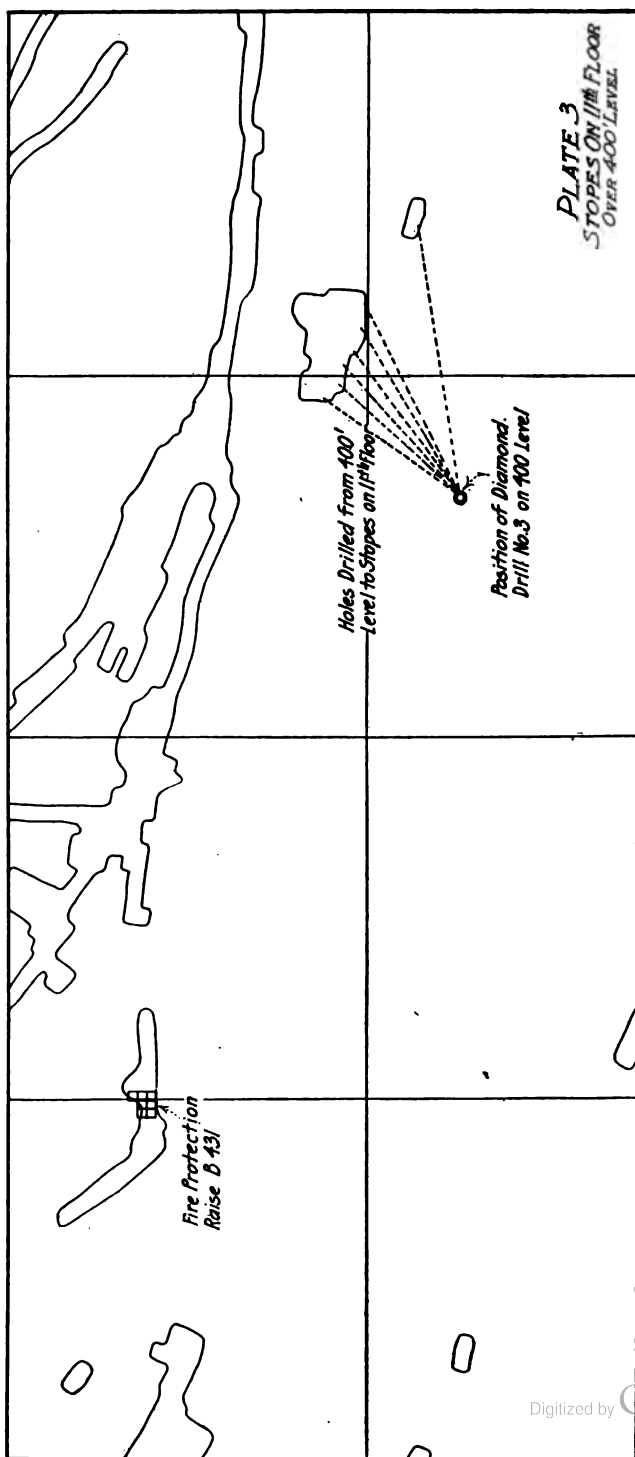
places, putting in new timber as we advanced. After an old drift had been opened in this way, we started at the end and came back, removing the wood posts and putting in old air pipes in their places, then filling the whole drift with waste. The work was arranged so that the waste from places we were timbering was banked in the places we were filling; 1,500 ft. of this was done, all by contract. The cost was high because: The original work necessitated spiling; 670 gal. of water a minute were running through these workings; the air was so poor that candles were useless and carbide and electric lights had to be used; and much time was lost in repairs to the tracks and pipe lines. For some time hot water made it impossible to work in some places. Some of the raises were concreted for a set below the sill after the timber had been removed. All the concrete was mixed by machine at the collar of the shaft. Plates 10 and 11 give a good idea of the extent of the work.

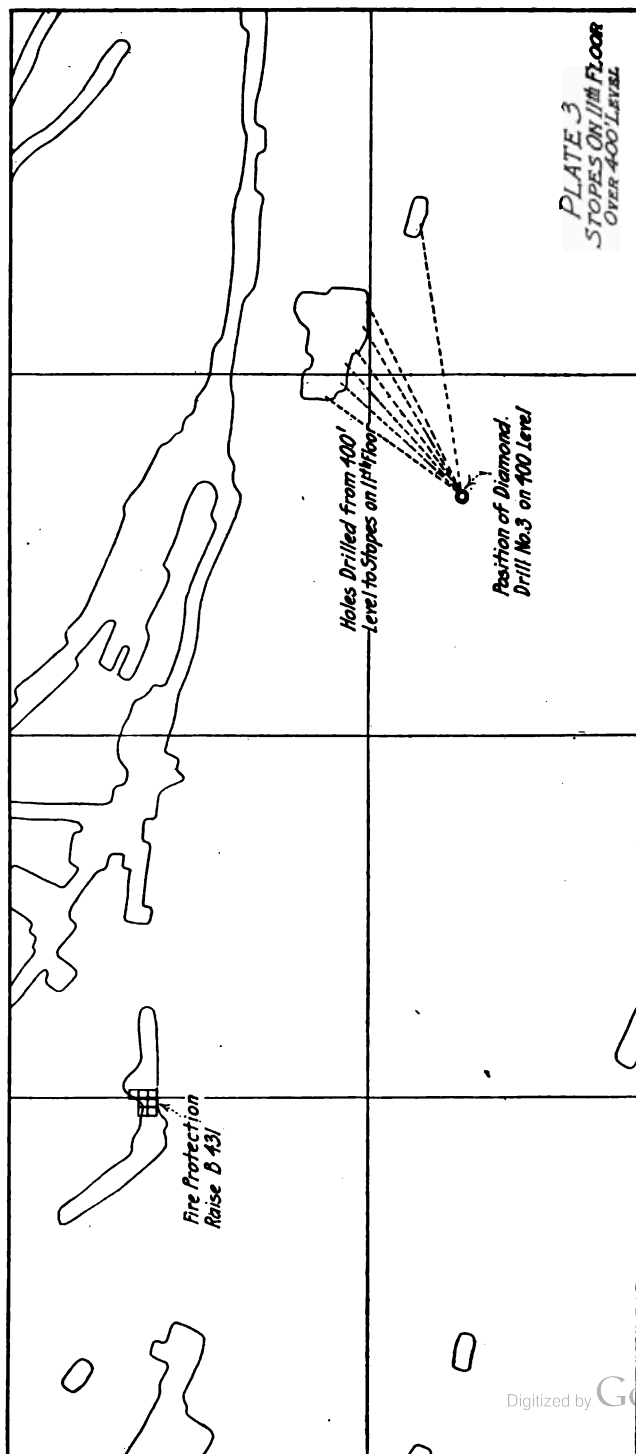
Fire-protection raises B-650 and B-651 were run through the gobs above to the 500 level. (See Plates 9 and 10.) All of the raises from the 600 to the 200 were three sets long east and west, only the two west sets of which were timbered. The west set was the manway, the center set was the chute, and the east set was filled with waste. When all were finished we had a wall of dirt filling from the 600 to the 200 which was fireproof but not gasproof. This we decided was sufficient if we should extinguish the fire. Had it been necessary, we decided to fill the chutes with concrete to make them gas and fire proof. Near the raises on each level we built permanent bulkheads with iron doors to cut off the fire zone from the rest of the mine.

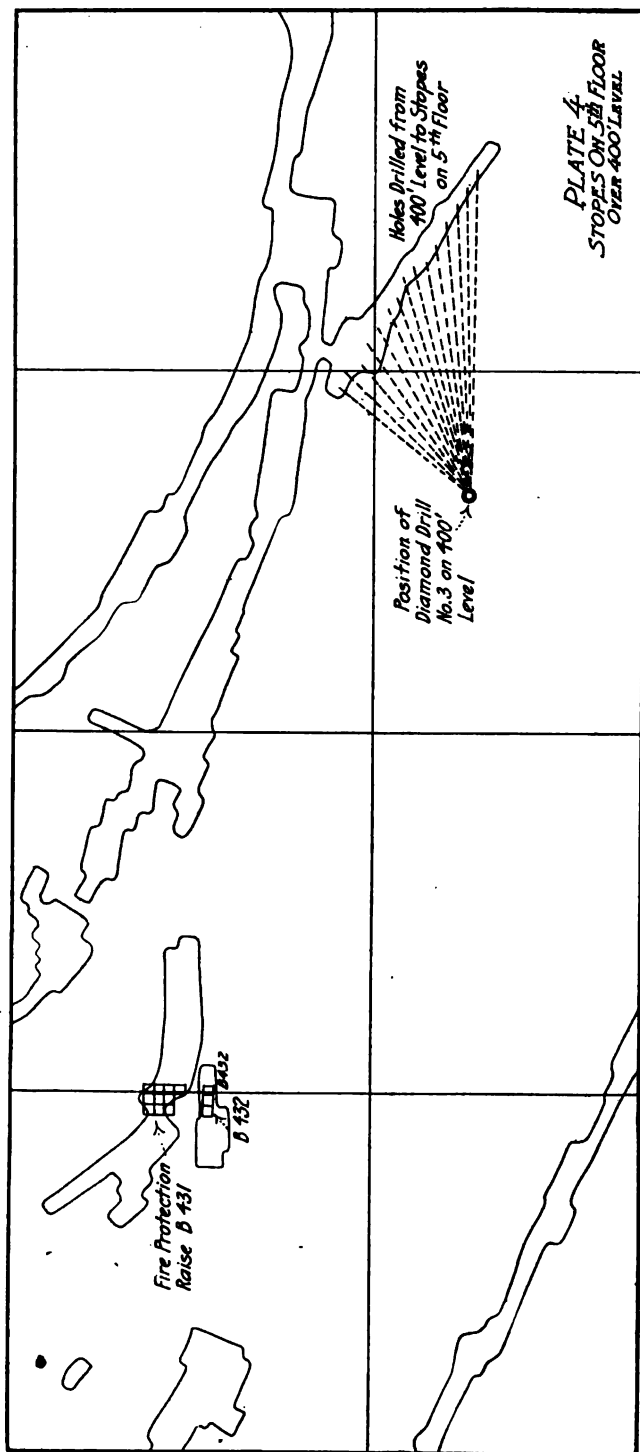
At the end of December, 1913, or six months after the fire started, we could say positively that there was no fire in the mine. As late as Dec. 1, 1914, this fire zone has been inspected and no trace of gas or smoke was evident.

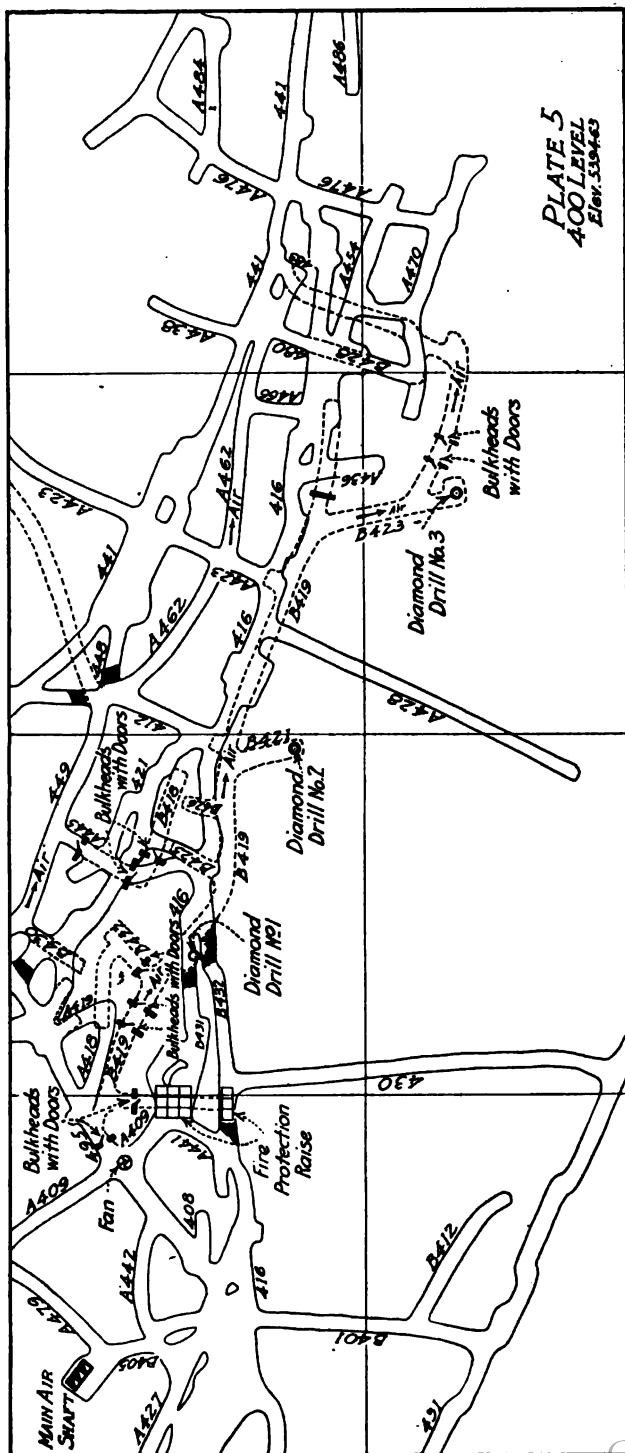


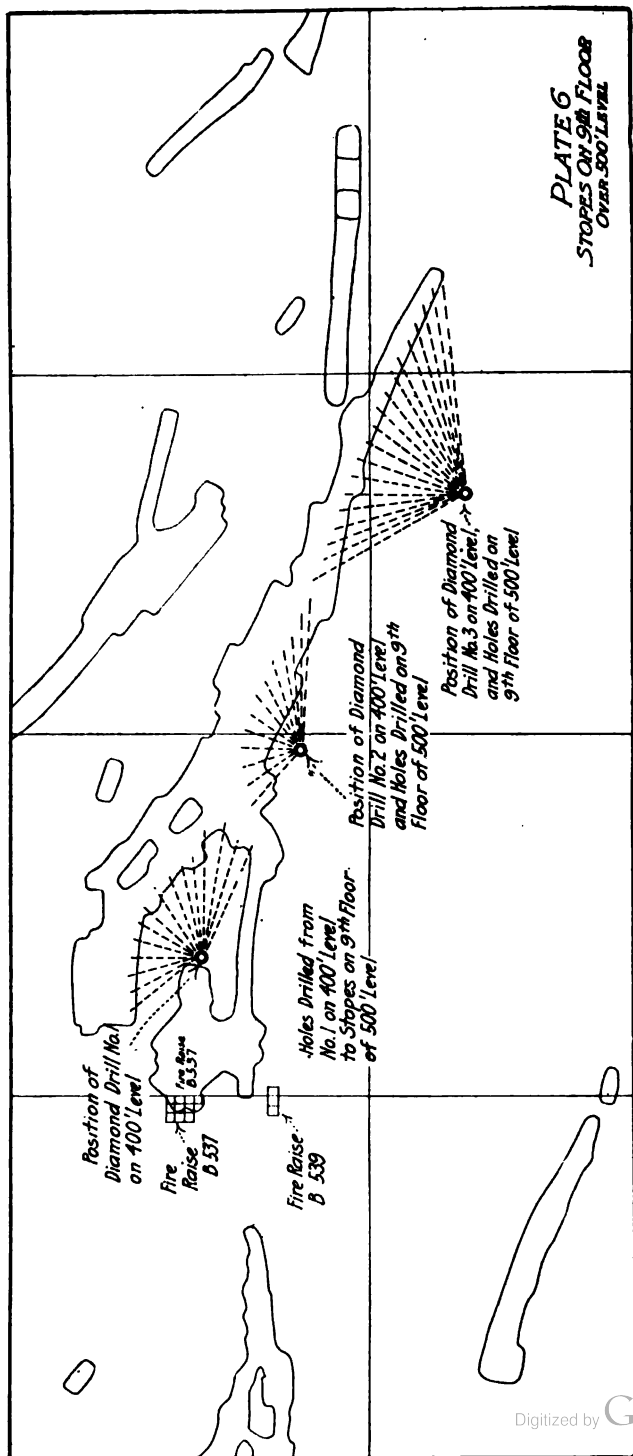


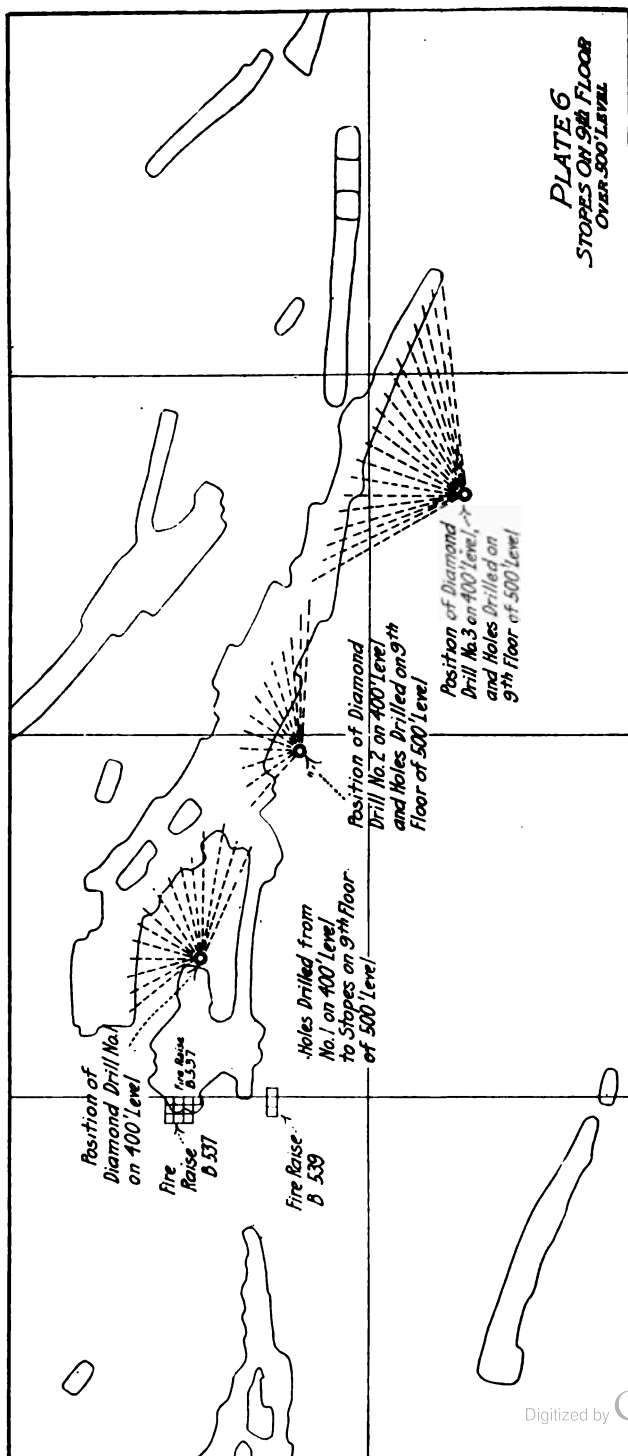


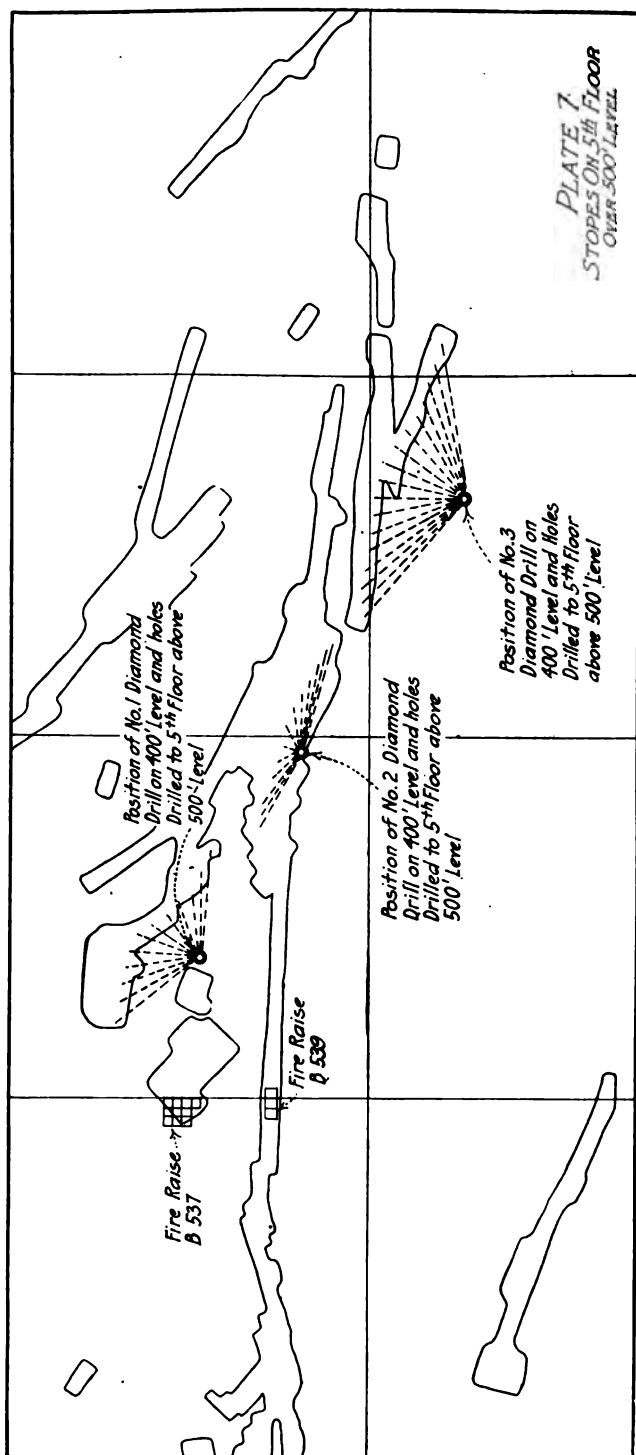


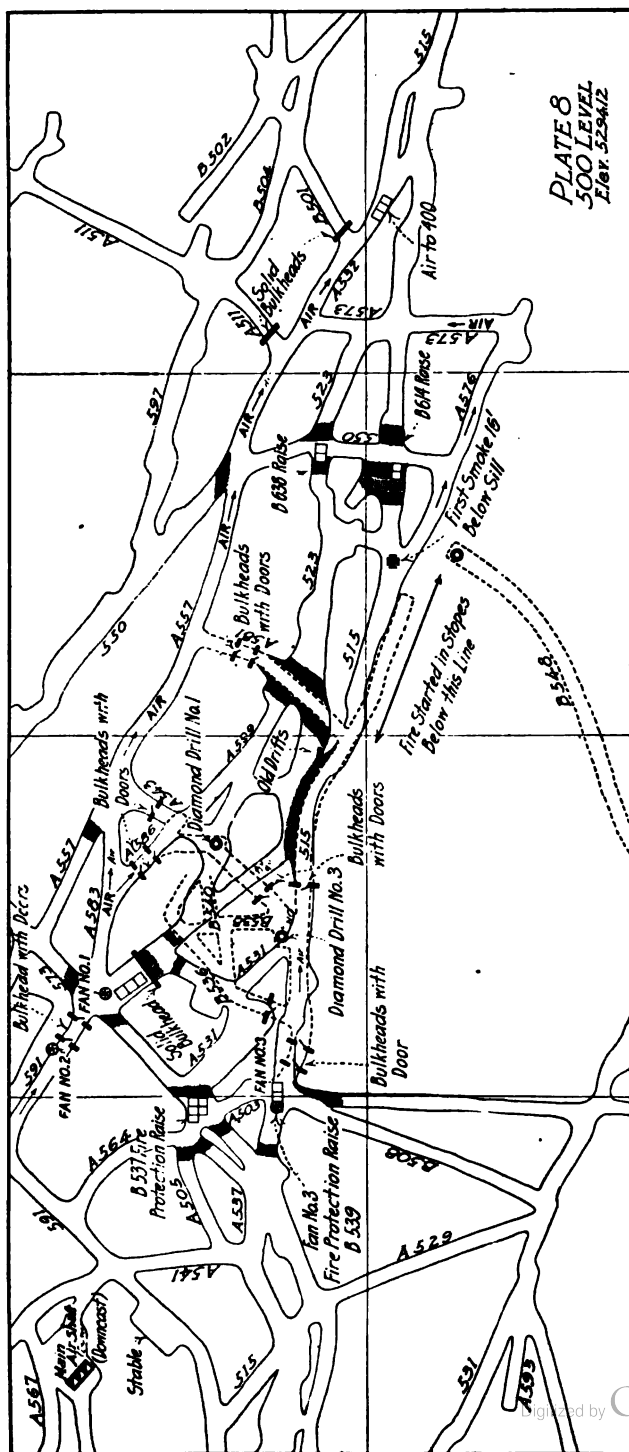


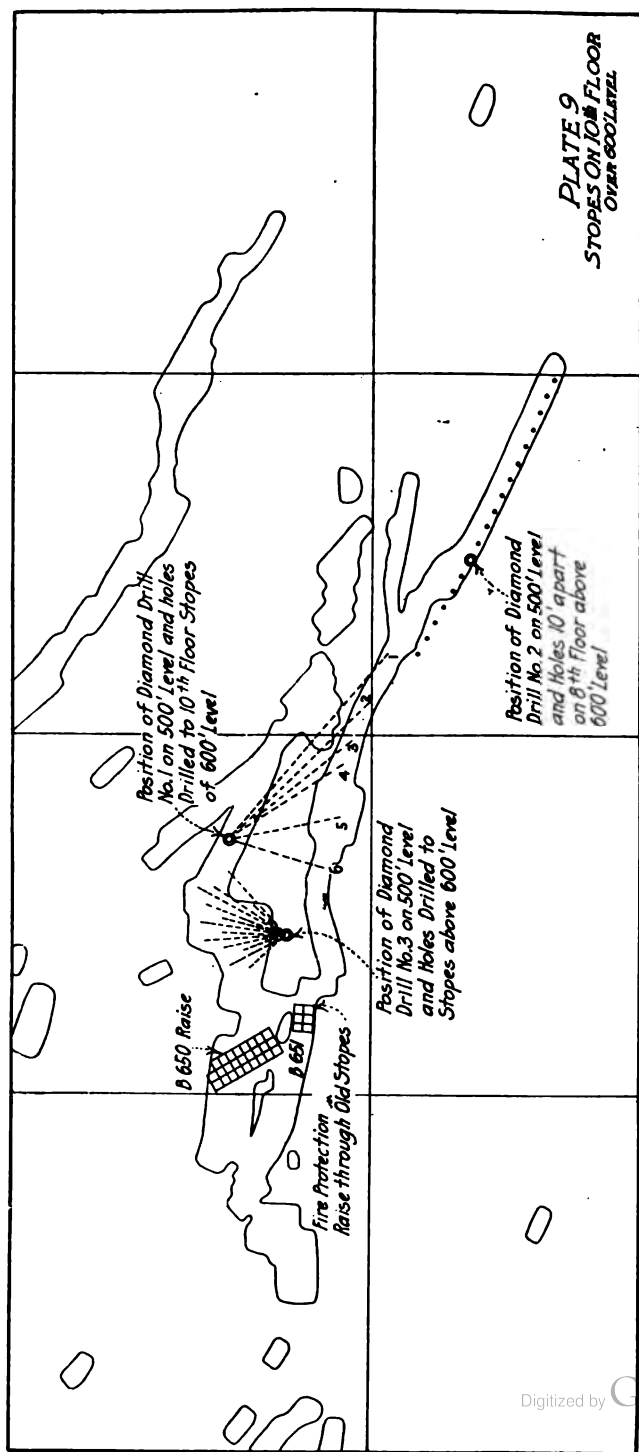


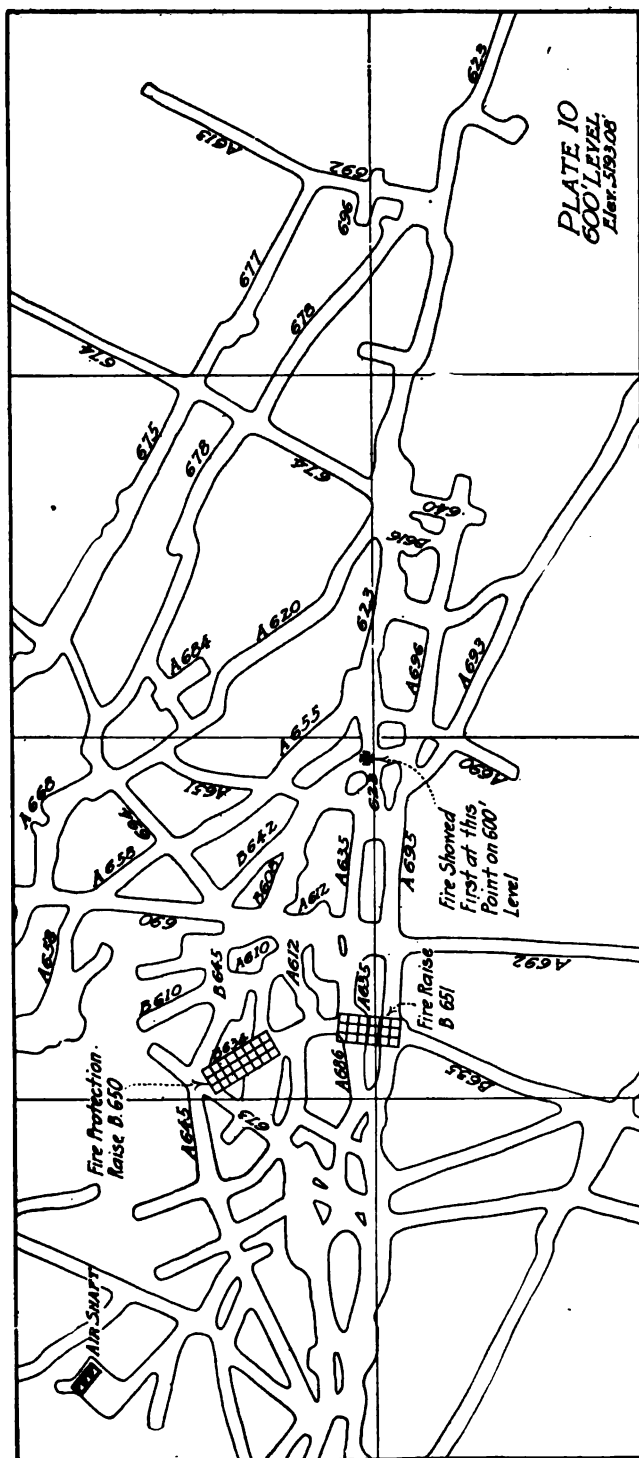


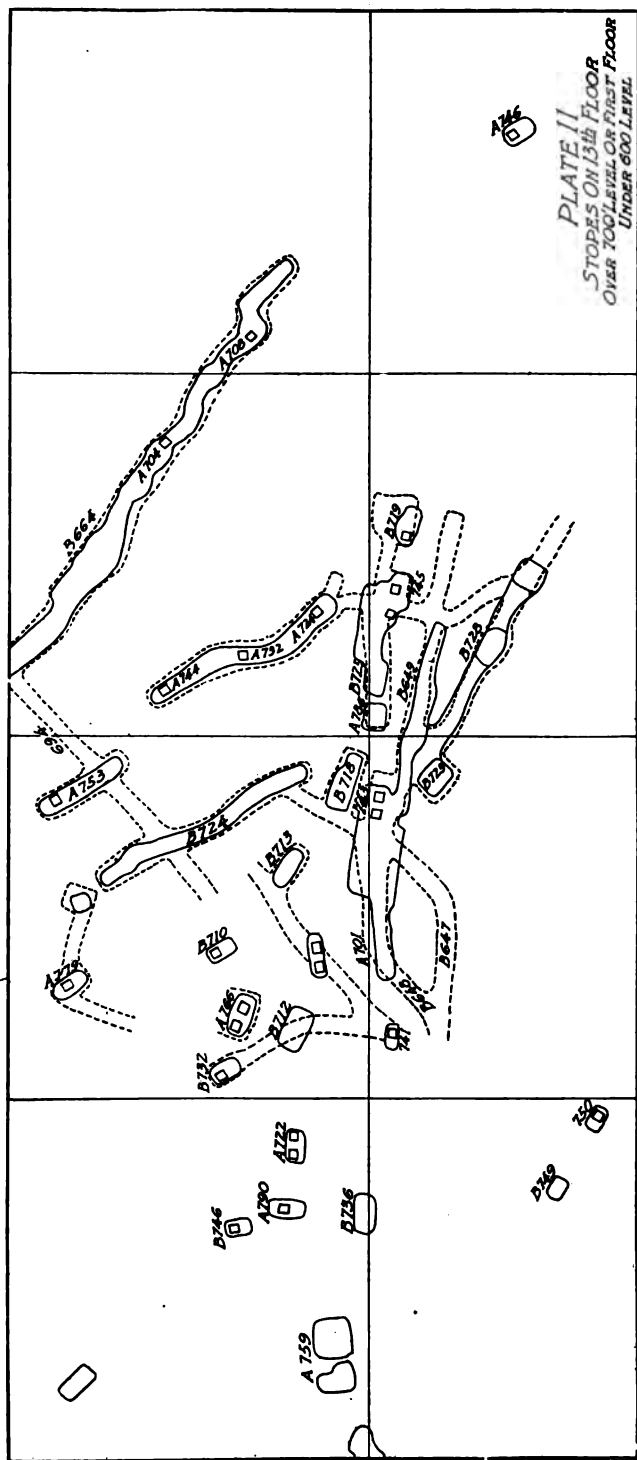












TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, Sept., 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

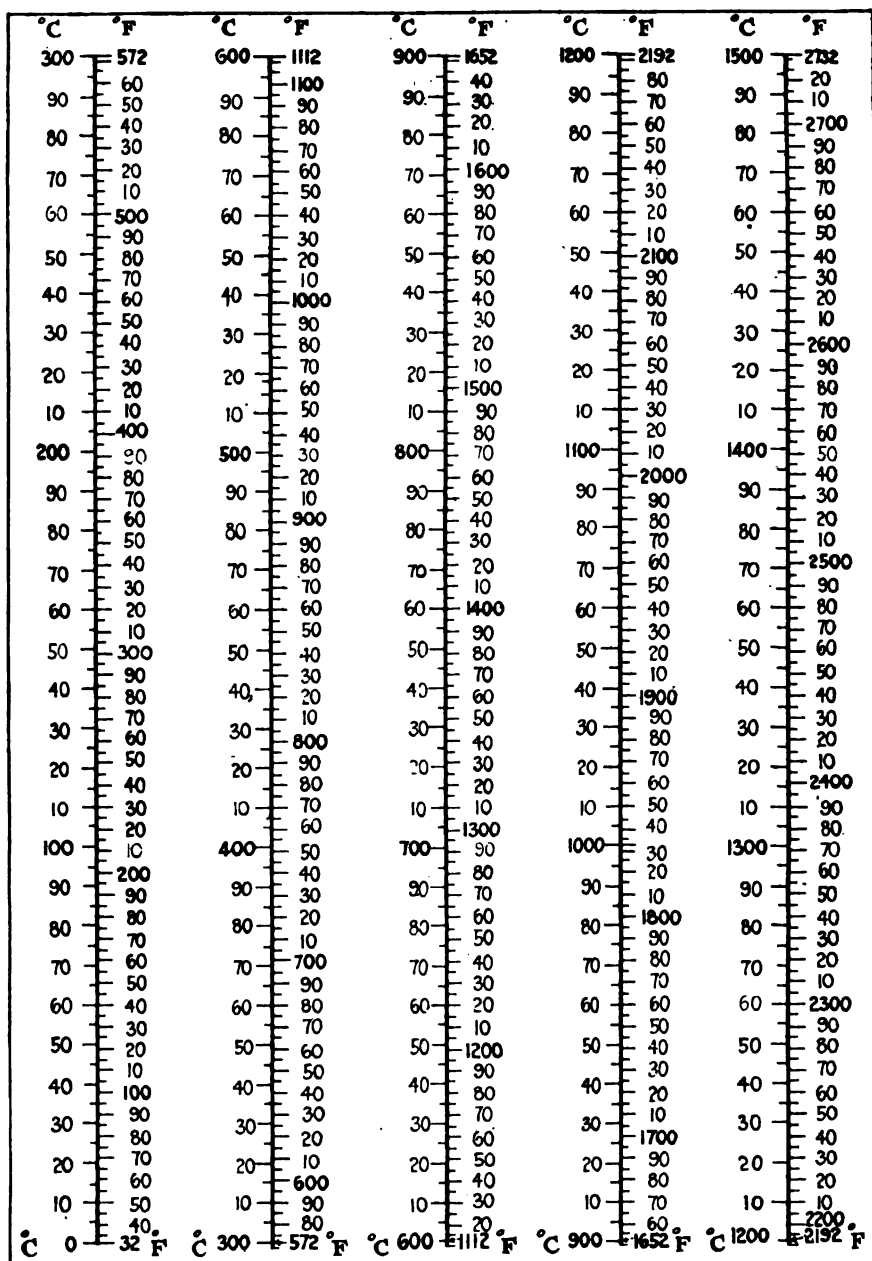
Conversion Scale for Centigrade and Fahrenheit Temperatures

BY HUGH P. TIEMANN,* B. S., A. M., PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

THE desirability of employing the centigrade scale for the measurement of temperatures is becoming more and more recognized in this country, particularly in view of the fact that this scale is used almost exclusively in technical papers and publications—exclusively in those published in French or German. There are, however, a large number of factories and laboratories which still employ the Fahrenheit scale. It therefore frequently becomes necessary to convert one into the other and the accompanying chart (Fig. 1) has been found very convenient for this purpose.

* Metallurgist, Carnegie Steel Co.



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Underground Mining Systems of Ray Consolidated Copper Co.

BY LESTER A. BLACKNER, M. E., RAY, ARIZ.

(San Francisco Meeting, September, 1915)

THE PROPERTY AND LOCATION

THE Ray Consolidated Copper Co.'s mining property is located on Mineral Creek 6 miles north of Kelvin, at Ray, Pinal County, Ariz. (Fig. 1).

The mining claims now owned by the company consist of 126 lode claims, comprising 2,144.9 acres, containing at the beginning of operations 82,904,368 tons of 2.19 per cent. copper ore.

GENERAL CHARACTERISTICS OF THE OREBODY

The main orebody is a disseminated deposit in schist and porphyry formed by secondary enrichment. Its existence was proved by churn drilling, and it is at present one of the largest proved copper deposits in the world. The orebody itself covers 205 acres with an average thickness of 121 ft., and occupies a definite belt which has a northwest direction. Its length is approximately 5,000 ft. The width varies, being about 2,000 ft. at the west end and 2,500 ft. at the east, narrowing down irregularly from both ends to a few feet in the center. The thickness of the body varies greatly along the line of lode, ranging from a few feet up to 470 ft. thick.

The ore horizon is not constant, but varies, following in a broad sense the topography. The body in general dips slightly to the northeast, and is broken up by numerous small faults and fractures. The ore-bearing formations, consisting of mineralized schist and mineralized granite porphyry, stand fairly well and offer no difficulty in mining operations.

The bulk of the ore is chalcocite disseminated in schist. In places cupriferous pyrite is closely associated with the chalcocite. The mineralized granite-porphyry formations are contiguous to the ore-bearing schist.

Overlying the orebody is an oxidized zone of leached iron-stained schist, averaging 252 ft. in thickness, termed "capping." The line of



FIG. 1.—MINE OF RAY CONSOLIDATED COPPER CO., RAY, ARIZ. TEAPOT MOUNTAIN IN THE DISTANCE.

demarcation between the ore and capping is easily discerned, owing to the difference in color. In a few places along this line small amounts of the carbonate and silicate of copper occur. Within the orebody a little native copper has been found, and at the base of the oxidized zone small areas contain cuprite.

The foot wall of the ore in many cases is terminated by a diabase intrusion, and in other cases fades off into *protore*, that is, material which by the continuation of the process of natural enrichment might be converted into ore. The ore along the diabase contact is of higher grade than the main orebody.

DETERMINATION OF TONNAGE AND VALUE

In arriving at the tonnage and copper content of the orebody every precaution was taken to secure accuracy. The developing was done with churn drills and underground drifting and raising. A complete survey of the property was made and the ground covered by a system of north-and-south and east-and-west co-ordinate lines 200 ft. apart. Churn-drill holes were put down as nearly as possible at the corners of the 200-ft. squares thus blocked out. The sampling was done very carefully. For every 5 ft. of drilling each hole was thoroughly cleaned and the sludge passed through a specially devised sampler. The samples were assayed locally and composites sent to outside laboratories for confirmation. Each drill hole was given a number, and in the office a sheet was kept showing in minute detail the geology, the men doing the work, time required, and assays for every 5 ft. of drilling. Records were also kept to determine the final cost per foot of drilling for each hole.

In determining the tons and assay value of the ore in any block the procedure was as follows: Each 200-ft. block was calculated separately. A survey of each block was made and the horizontal distances between holes determined. With these distances the actual area of the rectangle formed by the four holes was computed. After figuring the average assay for all the assays which could be called ore in a hole and multiplying it by the total number of feet of ore, the foot-percentage for the hole was obtained. Proceeding in a like manner the foot-percentage of each hole at the four corners of a block was found. Then after adding all the ore footages (ore in each hole) of the four holes together and dividing this total into all the foot-percentages added together the assay value in copper of the total tons of ore in the block was obtained; next by finding the average thickness of the ore in the block (thickness of the ore in each of the four holes added together and divided by four) and multiplying it by the area of the block and dividing by the number of cubic feet to a ton ($12\frac{1}{2}$) the total tons in a block was arrived at. Example:

A block having holes 251-249-267-263 for the four corners; area of the rectangle formed by these holes is 84,404 sq. ft.

Number of the Hole	Thickness of Ore, Feet	Average Assay of the Hole Copper, Per Cent	Foot-Per Cent.
251	90	1.77	159.30
249	345	1.86	641.70
267	100	2.65	265.00
263	80	2.62	209.60
Totals and averages	615	2.07	1,275.60

Average thickness of the ore in the block = $\frac{615}{4} = 153.75$

$\frac{84,404 \times 153.75}{12.5} = 1,038,169.20$ tons of 2.07 per cent. copper ore

By this method the tonnage and the assay value of each and every block of ore were determined. Then by adding the values for all the separate blocks together the total tonnage and the average assay for the entire orebody were obtained. As a check, the total tonnage was also figured by planimentering cross-sections of the orebody at 200-ft. intervals and computing the number of tons contained therein.

Of the 353 holes drilled to date, only 238 are inside the orebody and used in calculating the ore tonnage; they represent 106,971 ft. of drilling. It is gratifying to note that all the underground development and prospecting thus far have checked with accuracy the churn-drill results both as to assays and tonnage.

SYSTEMS OF MINING

Owing to the heavy overburden and the low grade of the ore, caving systems have been devised and adopted which consist in weakening a block of ore by means of a series of shrinkage stopes or "ore-filled rooms." Then after undermining and shattering the remaining pillars the ore is drawn systematically, the capping crushing and settling gradually over it. Throughout all the work at Ray two systems have been used: The sub-level or motor-haulage system, employed in thick uniform blocks of ore; and the hand-tramming system, used in the shallower portions. From the accompanying sketches it will be seen that the method of carrying up the stopes and the manway arrangement are practically the same in both systems, the essential difference being in the method of handling the ore.

The same conditions as exist at Ray had been previously encountered at the Boston mine of the Utah Copper Co., Bingham, Utah, and the systems of mining initiated at that place were subsequently chosen for Ray. In explaining the "Ray systems" now in use it is necessary to go over important changes and steps at Bingham prior to their final adoption at Ray.

SUB-LEVEL OR MOTOR-HAULAGE SYSTEM

At Boston Mine, Utah Copper Co.

During the experimental stages of the system stopes 20 ft. wide were carried up on 50-ft. centers, leaving pillars 30 ft. wide between stopes, as shown in Figs. 2A to 2D. In all the early operations at Ray as well as at Bingham the main difficulty was with manway connections or entrances to stopes and methods of undermining pillars. Raises at 100-ft. intervals were run up in the centers of alternate pillars and at points every 50 ft. up these raises small crosscuts were run out from the raise to the edges of the stope on either side, to provide manway connections or entrances to stopes. Men entering a stope passed up through the raises to the first crosscut above the broken muck in the stope. However, this means of entry did not prove satisfactory, as the pillars usually faulted or sloughed before the stope reached capping, cutting off the means of entry.

The next method was to run "manway drifts" parallel to the center line of stopes down the centers of the 30-ft. pillars on sub-levels 50 ft. apart, as shown in Fig. 2C, with crosscuts or entrances to stopes 100 ft. apart. In addition to this means of entry, "pole roads" 100 ft. apart were carried up along one side of a stope, these being located opposite the crosscuts on the sub-level. The pole roads consisted of slabs (that is, segments of round poles) 6 to 8 ft. long, placed horizontally, one directly over the other, 2 to 4 in. apart and in front of a groove or niche in the side wall of a stope. These poles kept the muck from entering the manway. The groove was cut into the wall deep enough to allow a man to pass through with comfort. In order to enter a stope by means of the pole road the men climbed up ladderways in the center of the pillar from the motor level to sub-level 2, then through a small crosscut to the pole road, then up the pole road to whatever level the stope may have reached (see Fig. 2D); or else they could walk down the manway drifts on either sub-level 3, 4, or 5, then through a crosscut to the pole road, and up through to the stope. Thus it can be seen that in theory the entries were convenient and plentiful. However, such an expensive network of manways did not work out in practice, as the pillar drifts, owing to crushing and faulting of the pillars, in places had to be timbered, thus becoming expensive to maintain. The pole roads also gave trouble, as too much muck had to be blasted into them in making the niche to be handled conveniently on sub-level 2, and at times the side walls around the niches broke wide, and expensive blocking and timbering were necessary to keep them open. At Ray, the pole roads were replaced with man-ways made of frame cribbing, so that the latter difficulty was overcome.

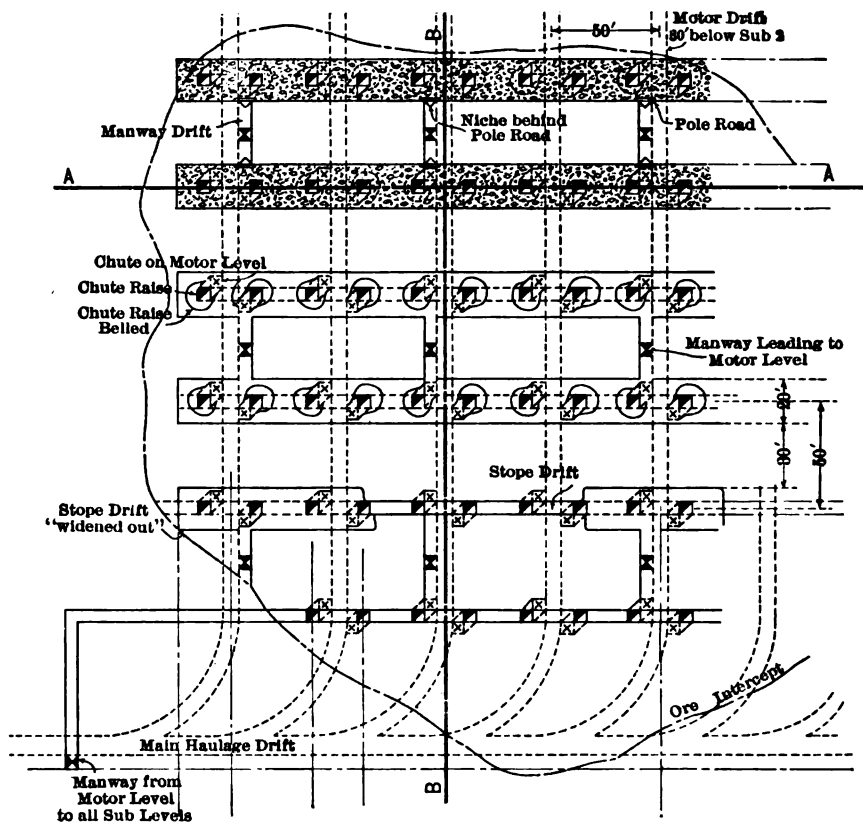


FIG. 2A.—SUB-LEVEL 2. MOTOR-HAULAGE DRIFTS ON MOTOR LEVEL DOTTED.

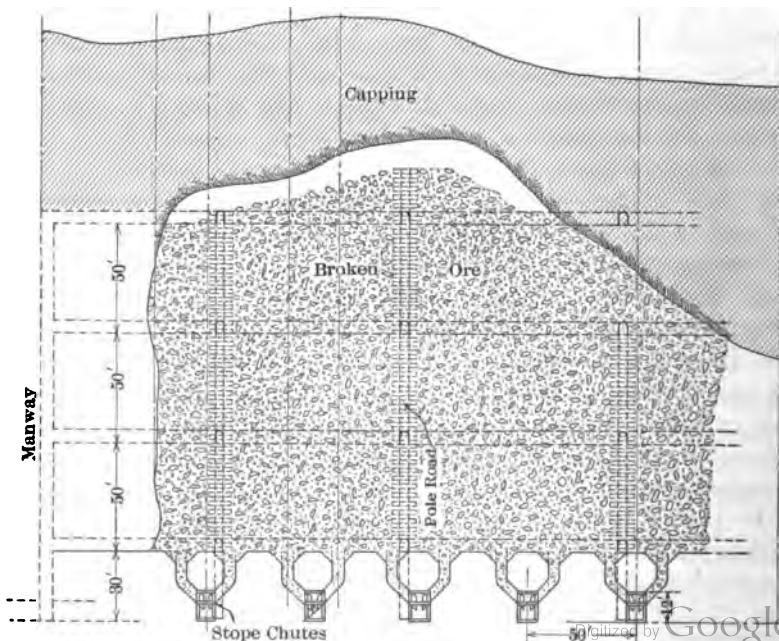


FIG. 2B.—SECTION A-A, SHOWING SUB-LEVELS AND POSITIONS OF POLE ROAD.

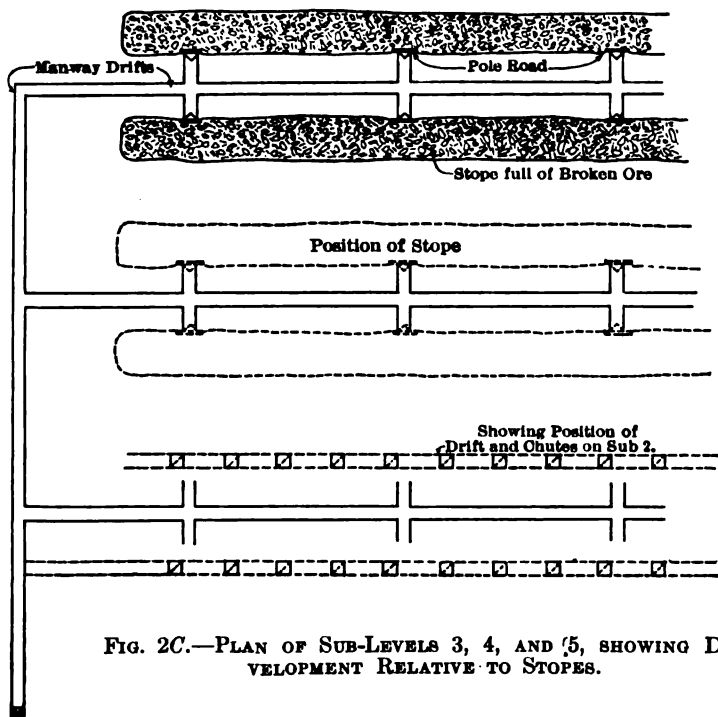


FIG. 2C.—PLAN OF SUB-LEVELS 3, 4, AND 5, SHOWING DEVELOPMENT RELATIVE TO STOPES.

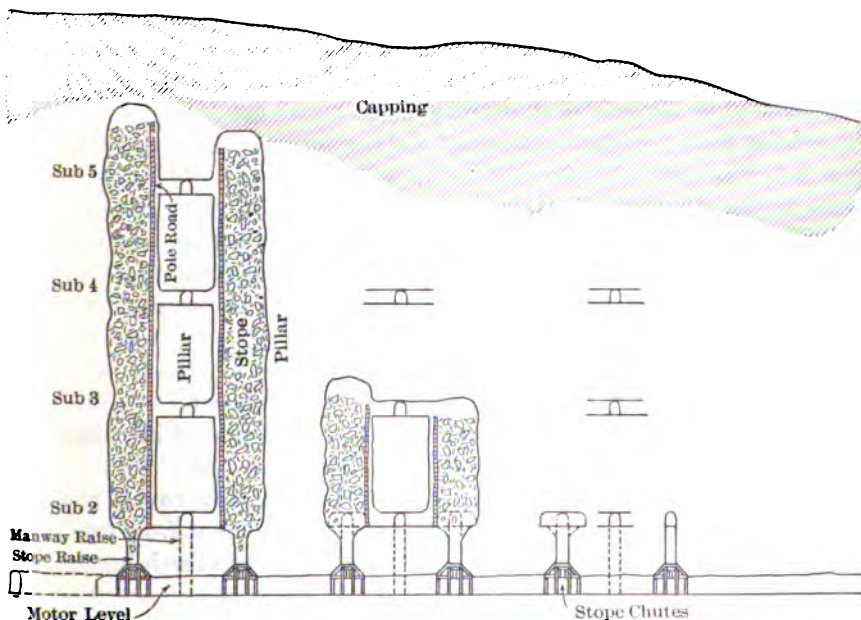


FIG. 2D.—CROSS-SECTION ALONG B-B, FIG. 2A.

FIGS. 2A TO 2D.—SYSTEM OF MINING USED AT THE BOSTON MINE, UTAH COPPER CO., BINGHAM, UTAH, AND IN EARLY OPERATIONS BY RAY CONSOLIDATED, AND FROM WHICH WAS DEVELOPED THE "RAY SYSTEM" OF UNDERGROUND MINING WITH MOTOR HAULAGE.

At Ray Consolidated Mine: Early Method

At Ray, the 30-ft. pillars, as explained in the foregoing, in portions of the orebody did not crush as was expected, and it was necessary to put up regular shrinkage stopes, termed "pillar stopes," 8 to 10 ft. in width, in the centers of the pillars. Then it was found that even the remaining pillars, 10 ft. in width, between the pillar stope and original stope, did not cave, and that an additional small stope had to be run up, perhaps 8 to 10 ft. From the above it can be seen that this method of blasting or undermining the 30-ft. pillars became expensive, awkward, and unsystematic, especially when it is remembered that manways had to be provided for the pillar stopes and all the above work done before a block of ground could be drawn as a reserve. In addition, the drawing-off chutes along the haulage drifts on the motor level were put in staggered fashion on 10-ft. centers with loading platform between them, and the raises from these chutes were run up vertically to the sides of the stope above instead of on an incline to the center. This was abandoned because the raises coming up on the sides of the stope made it difficult to draw the stopes evenly, thus hindering drilling operations in the stopes themselves.

In order to overcome these difficulties a decided change in the arrangement was adopted: All stopes to be 15 ft. wide and spaced 25-ft. centers instead of 50-ft. centers, leaving a pillar 10 ft. wide, which could easily be undermined and shattered; also to have but two sub-levels; the first sub-level 30 ft. above the motor level, and the second sub-level, manway or ventilation level, near the top of the orebody, or about 100 ft. above the first sub-level, and that the drifts on the second sub-level be run at right angles to the center line of the stopes instead of in the pillars or parallel to the center line of the stopes, as was the former practice, and that manways be provided by running up raises from the first sub-level to the second sub-level at intervals of 100 ft. along the stope.

At Ray Consolidated Mine: Present Method

To simplify the present system of mining with sub-level it is expedient to classify the work into different stages (Figs. 3A to 3D).

First Stage: Drifting on Motor Level.—The orebody covers such an area that it has been found more economical to mine it from two main shafts. There is, however, a third shaft at which the ore is high grade, and is being mined by the square-set method.

At shafts 1 and 2 the "Ray system" is being used and the mines are opened up by three motor-haulage levels. On each motor level a main drift (double track, 30-in. gauge) with two compartments (each 7 ft. wide, 8 ft. high, timbered with 12 by 12 in. posts, 12 by 14 in. caps,

6 by 12 in. sills, and 10 by 12 in. collar braces 4 ft. 2 in. long so as to place posts on 5-ft. centers) runs out from the hoisting shaft for a short distance to provide storage and passageways for trains and then narrows down to a single-track drift which is extended out along one side of the orebody. From this main drift a system of parallel drifts are turned off on a 60-ft. radius curve at intervals of 50 ft. and these are extended entirely through the orebody to a "fringe drift" which follows the ore intercept. Somewhere along the main drift or one of the parallel drifts in a convenient place outside the orebody a "permanent raise" is run up to the sub-levels. Such a raise is usually widened out and cribbed into two compartments (cribbing 6 by 10 in. by 9 ft. long, dapped 1½ in. deep and 6 in. back on the narrow sides of both ends and also in center for spreaders 6 by 10 in. by 5 ft. long. Spreaders are all dapped 1½ in. deep and 6 in. back on the narrow sides of both ends). One compartment is used as a manway (size 3 ft. 6 in. by 4 ft.), the other (size 4 ft. by 4 ft.) for hoisting supplies such as steel, pipe, hose, cribbing, powder, etc., for the stopes.

Second Stage: Chute Building on Motor Level.—Usually as the parallel motor drifts are driven they are timbered within the ore limits with 12 by 12 in. by 7 ft. 6 in. posts, dapped 1 in. deep on three sides at upper end to receive the cap and collar braces and 1 in. deep on drift side of bottom end to receive sill. Caps are 8 by 12 in., 9 ft. long, dapped 1 in. deep on all four sides at the ends and 11 in. back on three sides and 8 in. on the fourth or top side (on one of the 12-in. faces) to receive pony posts. The cap rests flatwise on the posts. Sills are 6 by 12 in. by 9 ft. long dapped 11 in. back and 1 in. deep on the upper side of both ends to receive posts. Collar braces are 8 by 12 in. by 4 ft. 2 in. long, framed 1 in. deep and 1 in. back on one 8-in. face at the ends to fit in with cap. The caps and collar braces are made of light timber to be subsequently protected with "pony sets" (Fig. 4).

Outside the ore intercept 12 by 14 in. timbers are used for caps and 10 by 12 in. for collar braces.

Within the ore limits below an area which is to be stoped, pony sets are erected on top of all drift sets and are made up of 8 by 12 in. by 5 ft. posts dapped 1 in. deep and 1 in. back on three sides at the ends; 12 by 14 in. caps 9 ft. long, framed 1 in. deep and 7 in. back on three sides of both ends to receive posts and collar braces, are used. Collar braces are 8 by 14 in. by 4 ft. 2 in. long, dapped 1 in. deep and 1 in. back on the drift side of both ends to fit in with the cap, thus spacing drift sets on 5-ft. centers.

In heavy ground a 6 by 12 in. filler block is placed under the cap, and supported in center of cap by 12 by 12 in. angle braces 5 ft. long over all, extending diagonally down to corners formed by pony posts and top of cap of drift set.

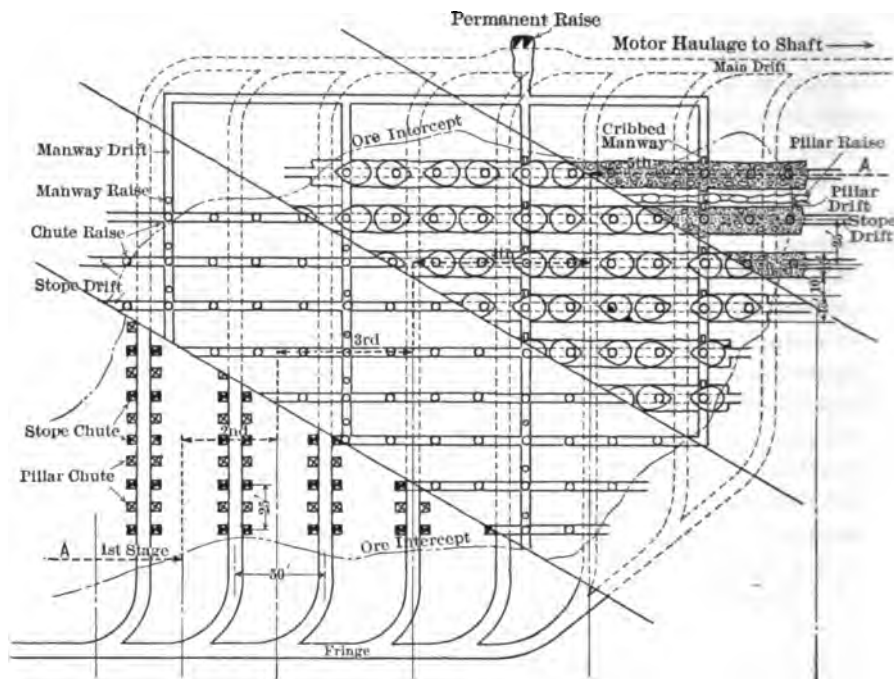


FIG. 3A.—PLAN SHOWING SIX STAGES IN DEVELOPMENT.

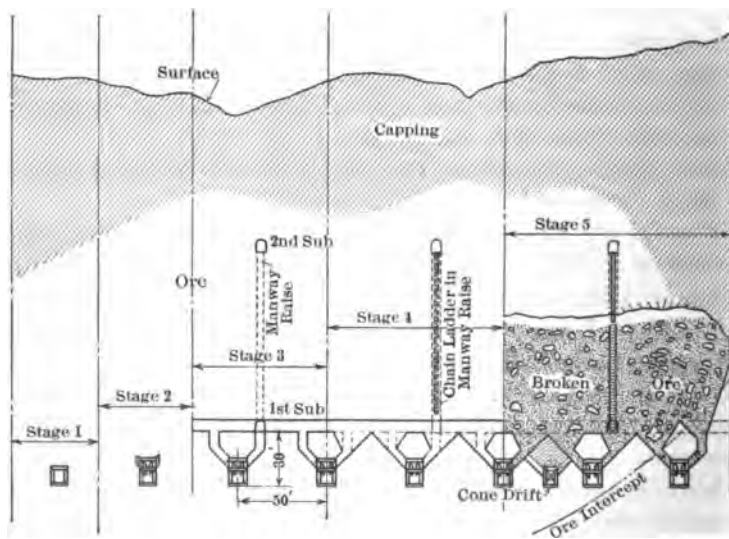
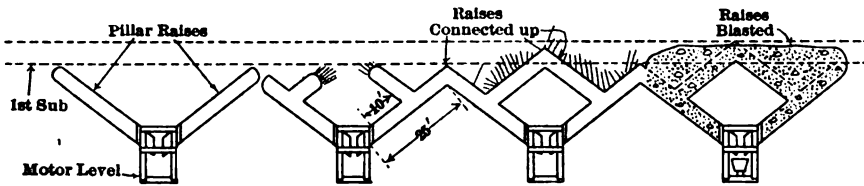
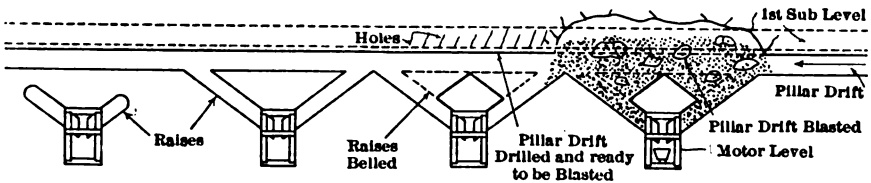


FIG. 3B.—LONGITUDINAL SECTION ALONG A-A, FIG. 3A.



Method No. 1.—Pillar Raises.



Method No. 2.—Pillar Drifts.

FIG. 3C.—METHODS OF UNDERMINING PILLARS. STAGE 6.

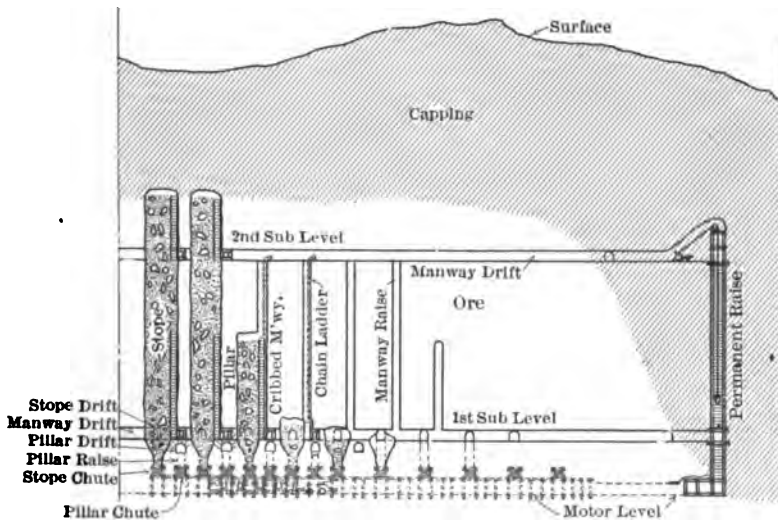


FIG. 3D.—CROSS-SECTION SHOWING SHRINKAGE STOPES AND PILLARS.

FIGS. 3A TO 3D.—“RAY SYSTEM” OF MINING. SHRINKAGE STOPES AND PILLAR CAVING WITH SUB-LEVEL MOTOR HAULAGE.

At intervals of 25 ft. along the drift, stope chutes are built opposite each other within a pony set so as to leave a 3-ft. clearance between them for muck to pass. Midway between, or $12\frac{1}{2}$ ft. from the stope chutes, pillar chutes are built. The chutes are made of 3 by 10 in. lumber, sides 3 ft. 6 in. long with 60° bevel on the two ends, and are nailed to pony posts. Bottoms are 3 ft. 6 in. long and nailed to the top of drift collar brace so as to have an incline of 30° for muck to run on. Chutes when finished are wide enough for a boulder 3 ft. in diameter to pass through (Fig. 5).

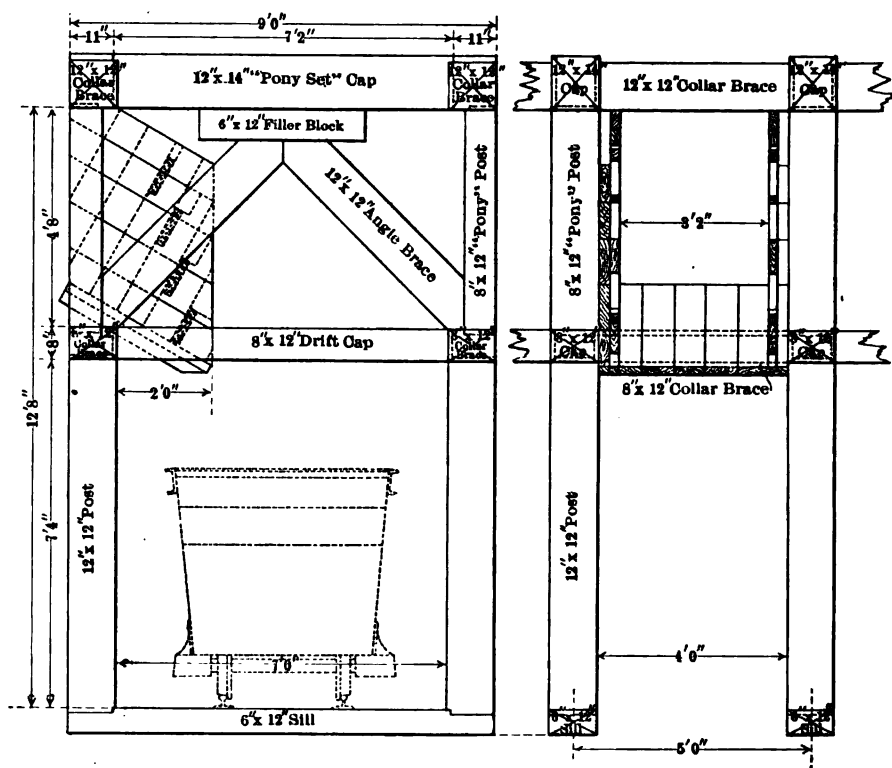


FIG. 4.—DRIFT TIMBERING FOR SUB-LEVEL, MOTOR-HAULAGE SYSTEM OF UNDERGROUND MINING.

Third Stage: Manway Drifts, Stope Drifts, and Chute Raises.—At the same time the motor drifts are being driven small manway drifts, size 5 by 7 ft., are driven parallel to them at intervals of 100 ft. on the first sub-level 30 ft. above. These manway drifts are offset $12\frac{1}{2}$ ft. to one side of the motor drift so as to be directly over a chute raise, and out of them at right angles are run a series of parallel stope drifts, spaced every 25 ft. over the entire orebody. These stope drifts are placed so as to be directly above stope chutes on the motor level, then when raises are run

up from the stope chutes they break into the stope drifts and become passages through which ore is drawn out into motor cars. The raises are usually 6 ft. in diameter and are run on an incline for 10 ft., then vertically until they hit the stope drift above.

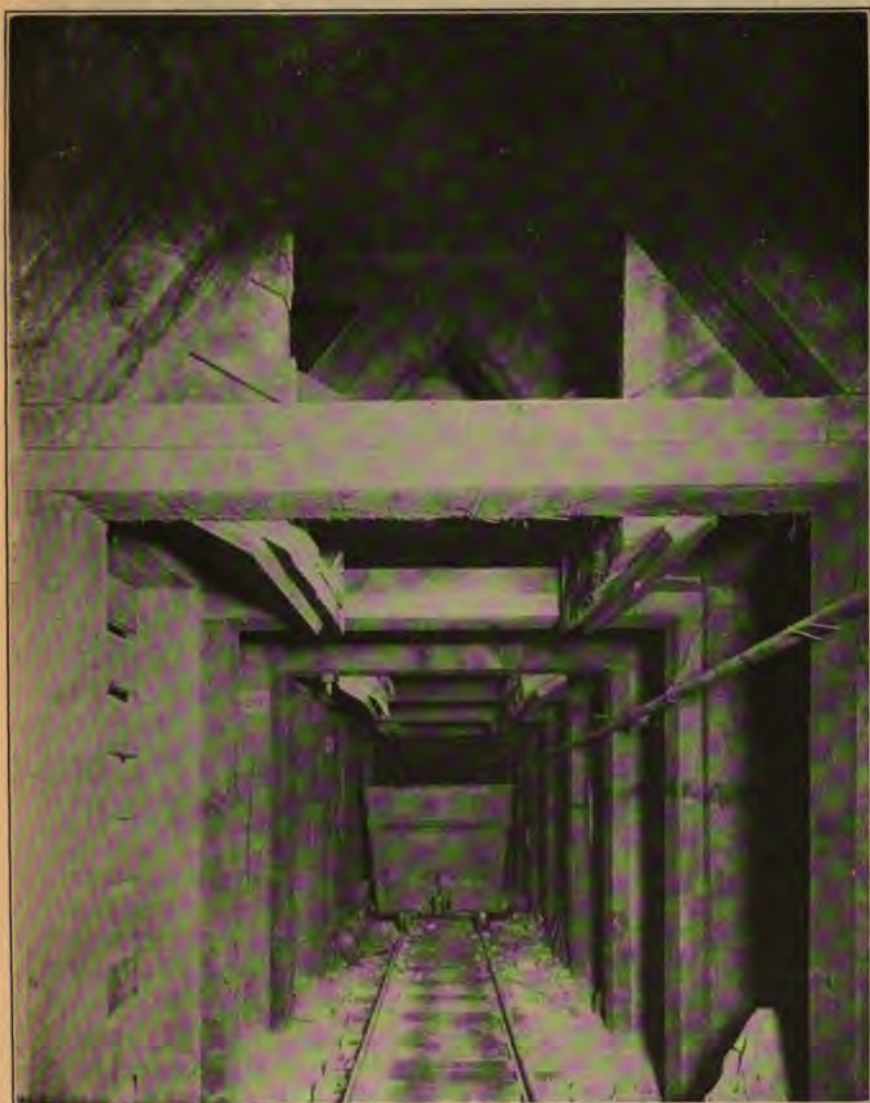


FIG. 5.—MOTOR DRIFT, SUB-LEVEL SYSTEM. SHOWING PONY SET, CHUTES, ANGLE BRACES, AND A 5-TON CAR.

On a second sub-level, which is usually about 100 ft. above the first sub-level, manway drifts are driven parallel to and directly over those

driven on the first sub-level. Then later when connections are made they serve for ventilation and passages through which men and supplies enter and leave stopes.

Fourth Stage: Manway Raises, Belling Out Chute Raises, Widening Out Stope Drifts, Building Manway Sets.—Along the manway drifts on the first sub-level, at intervals of 25 ft. and $7\frac{1}{2}$ ft. from the center of each stope drift, manway raises are run up to manway drifts on the second sub-level. Chain ladders are put into these raises so that men may descend or ascend from one sub-level to the other, and later the raises serve



FIG. 6.—INSIDE A STOPE. SHOWING CRIBBED MANWAY BELOW MANWAY RAISE.

as means of access to stopes from the second sub-level. While the manway raises are being run, men with stopper machines go down into the chute raises to “bell” them out, and when finished the chutes have the appearance of funnels or inverted bells.

In starting up a stope, men with stopper machines drill a line of holes all along and slanting into the sides of the stope drift. This line of holes when blasted, together with the work done to these drifts during “chute belling,” widens them out to 15 ft. so that they are ready to start up as “active stopes.”

As soon as the stopes are ready to start upward (Fig. 6), manway sets are erected in the manway drifts on the first sub-level. These sets

are built so that the 8 by 8 in. stringers on which the posts stand project out into the stopes, and a 3 by 3 ft. opening, made by cribbing (manway timbers) built up on these stringers, lies directly under a manway raise and serves as an entrance into an active stope from the first sub-level. The manway sets are made of 8 by 8 in. timbers, and the manway cribbing of 4 by 6 in. by 3 ft. 8 in. long, dapped 4 in. back and $\frac{3}{4}$ in. deep on the narrow side of both ends.

Fifth Stage: Mining Stopes to Capping.—In mining a stope (blasting down the back), two lines of holes, opposite each other, are drilled all along the back on both sides of the stope. The one line is drilled close to the side wall, with the holes slightly toeing into it; the other line, usually placed 4 ft. away from the side wall, is drilled so that the holes incline slightly toward the center of the stope. If the ground is very hard a third line is drilled still nearer the stope center. The machinemen working in the stopes use stoper machines and drill from 12 to 18 6-ft. holes each, per shift. These holes are usually loaded with four sticks of 30 or 40 per cent. powder; 6 and 7 ft. fuses are used, and each man blasts his own holes, going off shift. The oncoming shift draws off through the stope chutes into 5-ton motor cars on the motor level the excess ore, which is usually 33 per cent. of the total broken per round, thus giving the men plenty of head-room to re-drill the back while standing on broken ore.

The manways are built up with cribbing, when necessary, so as always to be open and above the muck. The machinemen in a stope receive air for their machines through a $1\frac{3}{4}$ -in. supply hose dropped down from the second sub-level through each of the manway raises. When a stope has reached a place about midway between the first and second sub-levels, the men naturally descend into the stope from the second sub-level through the manway raises instead of climbing up through the cribbed manways from the first sub-level. The cribbed manways become therefore a needless expense and are covered over the top with 2-in. lagging and left behind.

In hard ground the stopes are carried up 15 to 20 ft. wide, and in soft, "sloughing" ore 10 to 15 ft. wide.

Blacksmith shops are provided underground near the areas being stoped, so that men are quickly supplied with sharp steel and tools (Fig. 7).

Sixth Stage: Undermining Pillars.—After a block of ore has been weakened by a series of stopes the only step remaining in order for the entire block to crush and break is to undermine the pillars. Two methods are now in use:

Method 1: Pillar Raises.—Used in blocks of ore where the ground is hard and the stopes are carried up 15 to 20 ft. wide, leaving narrow pillars which are readily undermined. Starting with the pillar nearest the "fringe drift" on the motor level, raises are run out on flat inclines from

each of the chutes for the same pillar in the various drifts until they intersect each other. Then out of these raises at a distance 10 to 12 ft. from the chutes, raises are run back so as to connect with each other directly over motor drifts. After the raises have been connected up all along a pillar they are widened out, lined with deep holes and blasted so as to undercut the entire pillar completely. Proceeding in a like manner with each consecutive pillar, the whole block is finally undermined.

Method 2: Pillar Drifts.—Used in blocks of ore when the ground is sloughing, badly faulted, and packs, so that for safety to the miners the original stopes had to be carried up narrow, 10 to 15 ft. wide.



FIG. 7.—UNDERGROUND BLACKSMITH SHOP. SHOWING STEEL SHARPENER TO THE RIGHT, AND HAND FORGE TO THE LEFT.

In the center of and all along a pillar between two stopes, at a height of 22 ft. above the sill of motor level, small drifts are driven with plugger machines. Raises from the "pillar chutes" are run up on flat inclines until they intersect the drift. The drifts can be widened out as much as desired with stopper machines, to undercut the pillar. The raises are "belled," and the back of drift filled with deep holes and blasted so as to shatter the pillar completely.

The advantage of Method 2 is that a pillar of any width may be undercut and shattered; however, Method 1 is cheaper, and therefore is used where the pillars are narrow.

Seventh Stage: Reserve Drawing.—In most cases the orebody and capping, especially at No. 1 shaft, are badly shattered and broken up by small fracture seams so that when the pillars in any stoped area are undercut and the ore drawn, the capping directly over the outer edge of the area breaks in a nearly perpendicular plane to the surface.

In areas where the haulage drifts are to be abandoned as soon as all ore has been drawn, the procedure at present is to get the waste on a slight incline dipping toward the fringe drift. This is accomplished by drawing the chutes nearest the fringe more rapidly than those farther away, so that by the time the drifts take weight all ore will have been drawn and the occasional expense of retimbering, when all chutes in an area are drawn evenly, overcome.

An accurate account is kept of the ore drawn from every chute so that the ore remaining in each is always known. Only a few cars are drawn at a time, thus permitting the ore to settle gradually, with capping following after, thereby avoiding as much as possible the intermixing of the two.

For drawing chutes, empty cars are fed by the motorman through the fringe drift into the back ends of the parallel drifts, in which chutes are being drawn. These cars are then taken one by one ahead by a car pusher to the chute from which the ore is to be pulled. A chute blaster or loader stationed on a platform at the base of a pony set then lifts the boards or gates in the chute and allows the ore to run into the car. When a car is loaded a second car pusher shoves it down toward the main drift while his partner is spotting another empty for the chute blaster to load. All drifts are run on $\frac{1}{4}$ per cent. grade in favor of the load.

As soon as a train (usually 8 to 12 cars) has been loaded and pushed ahead, a motor comes through the main drift into the lateral; the motor helper couples the cars, and the motor then takes the train of cars to the shaft or tippie, where it is dumped, and the empty cars are returned to the back end of whichever drift may be in need of them.

The highest efficiency and best results are obtained when only two car pushers and one chute blaster (car loader) are used in each drift. Recently at the No. 1 shaft, six car pushers, three chute blasters, two machinemen, one mucker, and one chute repairer (timberman), working in four different drifts, loaded 400 motor cars, or an average of 150 tons per man, during a shift of 8 hr.

Machinemen drill and blast any large boulders which may lodge in a chute, also connect over raises preparatory for undermining pillars. The chute repairer replaces any broken chute boards, and makes other small repairs needed after blasting. The only men employed in a reserve-drawing section, other than those mentioned above, are timbermen on general repair work, a car checker, and a boss.

Eighth Stage: Cone Drifting.—After all chutes in a block have been drawn to capping, the next step is to recover the ore below the sub-level in the pillars between the motor drifts. This is done by first driving a small timbered drift parallel to the motor drifts in the centers of the pillars throughout the entire length of the block. Chutes are built opposite each other in every set along the drifts. A small stope extending along and directly over the drift is widened out and carried up to the sill of the first sub-level. Ore is drawn out into small cars (capacity 22 cu. ft.), which are then pushed by hand to a dumping chute, or winze, at the end of drift, where the ore is dumped to the next motor-haulage level below. When all chutes in this center cone drift have been drawn to capping, the small pillars still remaining on either side are taken out by slicing, so that eventually all ore above the sill of the motor-haulage level will have been extracted.

HAND-TRAMMING SYSTEM

This system differs from the sub-level method in that the stopes are started immediately above the drift sets and the ore is drawn out into cars of 22 cu. ft. capacity which are pushed by hand to dumping chutes.

At Boston Mine, Utah Copper Co.

Drawing-Off Chutes on One Side of Stope.—In the early experimental stages this method of stoping was very crude, as can be seen from Fig. 8A; on the tramming level a stope drift was run at right angles from a series of parallel drifts which had been spaced 30 ft. centers. The stope drift was widened out to 20 ft. by slabbing off the sides with "water-Leyner" drifters. At intervals of 60 ft. along the stope pole roads were provided for men and supplies to enter and leave stope. These pole roads were made by building up slabs or round poles in front of niches in the side wall of the stope. At first the excess ore broken by the machinemen was drawn off below by men shoveling it into cars holding 28 cu. ft.; this was soon found to be expensive stoping, so chutes were built in the faces of each drift, and ore then drawn direct from stope through the chutes into cars.

The disadvantages were that only a small tonnage could be obtained, as only one stope could be worked at a time; the stope being drawn on only one side made it difficult for the machinemen to work; cars had to be loaded at the ends instead of at the side, and consequently were continually over-running, necessitating shoveling to clean track after each tram; low final extraction, as ore in back side of stope could not be recovered, and as can be seen from the present Ray method of starting stopes, the work of drifting and slabbing on the sill floor was a waste of time and

money, as the void made by this work was refilled with unrecoverable broken ore from the stope.

Drawing-Off Chutes on Both Sides of Stope.—In order to overcome some of these defects the scheme shown in Fig. 8B was adopted. Here the tramming drifts, 60 ft. apart, were run down the centers of the pillars between stopes, and at intervals of 30 ft. crosscuts were turned off from them at right angles to the stope on either side. This arrangement provided chutes 30 ft. apart on both sides and all along each stope, so that the ore in the stope could be drawn down more evenly and a higher final extraction obtained; this also made it possible to work two or more stopes at a time, giving a greater output. However, the stopes were started in much the same way as before in that a small drift was run down the center position of a stope, later slabbed out to 20 ft., or to whatever width it was decided to make the stope, and the stope started direct from the sill floor of the tramming level. The pole roads for manways were run up in much the same way as previously explained. The disadvantages of loading the cars at the end; the poor ventilation in stopes; difficulty with manways; waste of time and labor of slabbing on sill floor before a stope could be started, still continued. Also, by reason of the fact that the crosscuts leading from tramming drifts were directly opposite each other, too large an opening was necessary, so that pillars took weight, making it difficult to keep tramming drifts open. The arrangement of having the chutes along the sides of the stope made it necessary for the machinemen at times to work out in the center where muck was continually falling (sloughing) from the back, and also made it necessary while starting up a stope to shovel the ore along the center, over to the sides above a chute.

Drawing-Off Chutes in Center of Stope.—In the next block of ore, to further reduce difficulties, the tramming drifts were run down the centers of every alternate pillar, thus eliminating one-half of the development work, and instead of turning the crosscuts to the stopes out of them at positions opposite each other they were placed staggered fashion (see Fig. 8C) so that the smallest opening possible was obtained at every point along the tramming drift. The crosscuts were extended in each case beyond the center line of stopes and two chute sets erected in each to provide chutes directly under the stopes instead of at sides. In starting a stope, raises were run up on flat inclines out of each chute, until they intersected, thus forming "hog-backs" between each pair of crosscuts. This method of starting stopes was an important change over the previous procedure, as it eliminated all the expensive drifting and slabbing on the sill floor under a stope, and made it possible to recover all ore broken in a stope. Manways were provided at first in much the same way as in early operations with the sub-level system as previously explained; i.e., raises were run up out of the tramming drifts at spaces

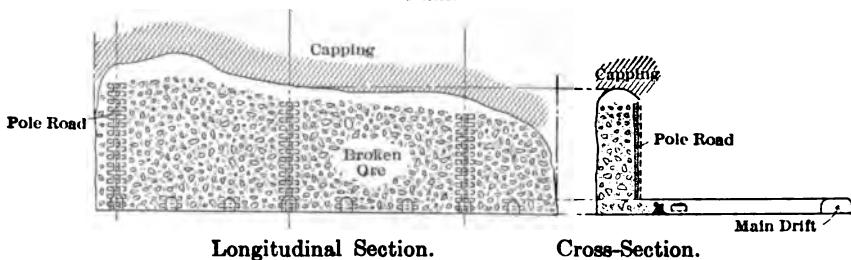
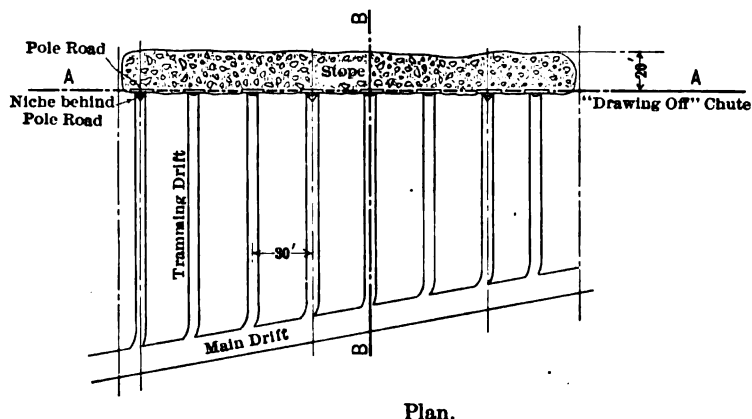


FIG. 8A.—SYSTEM OF MINING AT BOSTON MINE WITH DRAWING-OFF CHUTES ON ONE SIDE OF STOPE.

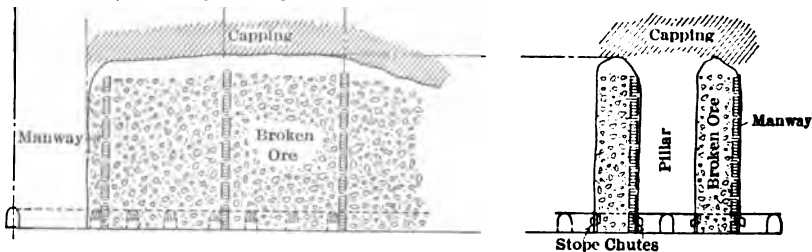
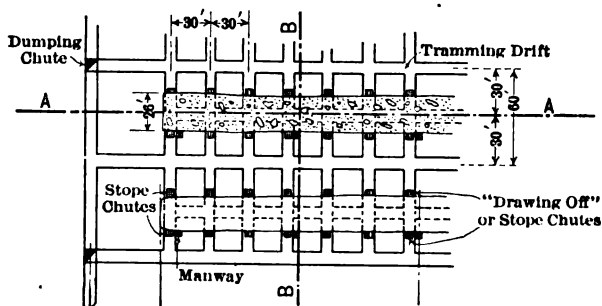
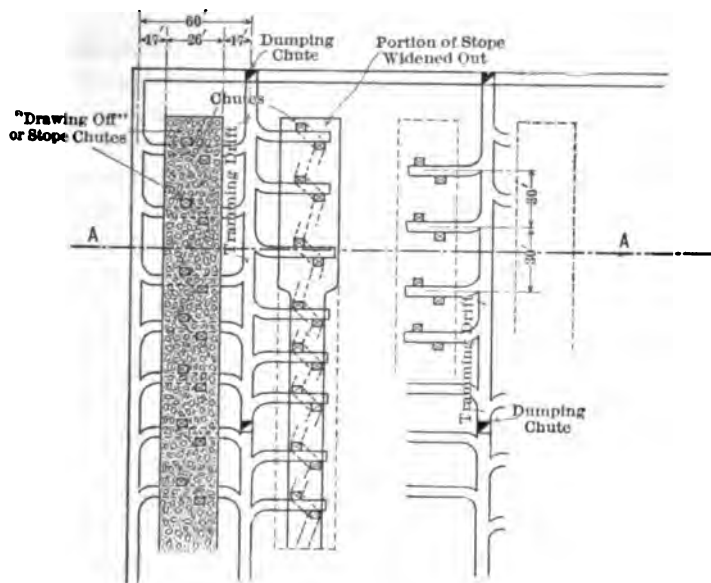
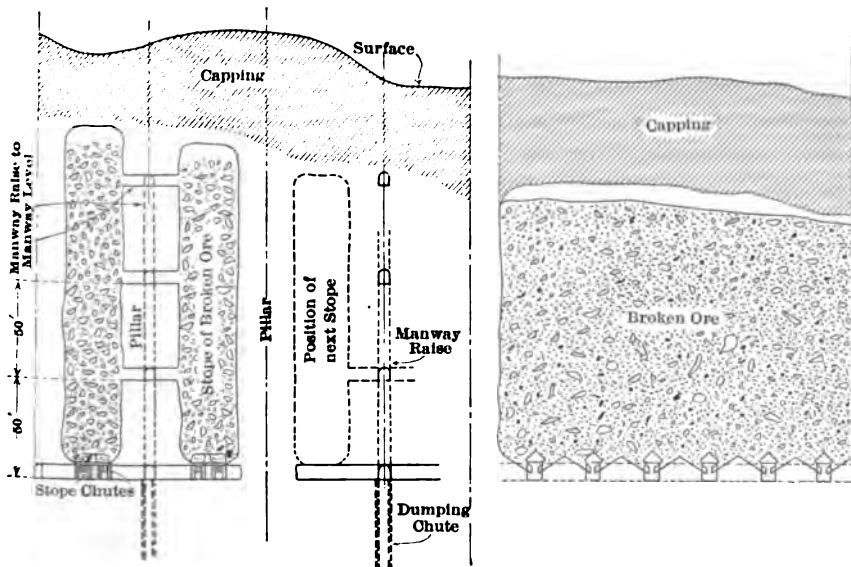


FIG. 8B.—SYSTEM OF MINING AT BOSTON MINE WITH DRAWING-OFF CHUTES ON BOTH SIDES OF STOPE AND TRAMMING DRIFTS IN EVERY PILLAR.



Plan of Tramming Level.



Cross-Section along A-A.

Longitudinal Section of Stope.

FIG. 8C.—SYSTEM OF MINING AT BOSTON MINE WITH DRAWING-OFF CHUTES IN CENTER OF STOPE AND TRAMMING DRIFTS IN ALTERNATE PILLARS.

60 ft. apart, and at distances of 50 ft. up these raises small crosscuts were driven over to the sides of stope to provide means for entering and leaving. These raises proved unsatisfactory and were abandoned, as the pillars became cracked and faulted, making them dangerous for the men to use, and necessitated driving drifts down the centers of alternate pillars between two stopes on the sub-levels, and out of them the crosscuts driven to hit the sides of stopes. These manway drifts proved satisfactory in the hard, uniform sections of ore, but when working in ground full of seams and faults they not only became impassable, due to breaking up of pillars, but the tramping drifts and crosscuts on the main level had to be held open by expensive timbering and retimbering. Such a condition obtaining at a time when active stoping was being done, it can easily be seen that heavy expense would naturally follow during reserve drawing of such blocks. In addition, the dumping chutes were poorly arranged for obtaining good results. Owing to the great distance to be traversed to dump their cars, the trammers unavoidably lost much time waiting their turns, consequently the output in tons per man was low as compared with results now being obtained at Ray. If reserve drawing had been attempted with such a system it would have been impossible to extract the ore in the pillars without driving additional drifts.

At Ray Consolidated Mine

Preparatory Work.—The system now in use is shown in Figs. 9A to 9D, and consists essentially as follows: All the tramping drifts are at right angles to the stopes, instead of parallel to them. Along two opposite sides of a block of ore fringe drifts are driven. Between these two drifts a system of parallel drifts on 25-ft. centers are driven (track 18-in. gauge, 12-lb. rail). They are timbered with 12 by 12 in. posts 8 ft. long, dapped 1 in. deep and 1 in. back on three sides of the upper end to receive cap and collar braces and the same on one side of the bottom end to receive the sill. Caps are 12 by 12 in. by 6 ft. 10 in. overall, framed 1 in. deep and 11 in. back on three sides of both ends to fit on posts and receive collar braces. Collar braces are 10 by 12 in. by 4 ft. 2 in. overall, framed 1 in. deep and 1 in. back on one side of the two ends to fit in with cap, placing posts on 5-ft. centers. Sills are 6 by 12 in. by 6 ft. 10 in. overall, framed 1 in. deep and 11 in. back on the top side of both ends to receive posts. In sections where the ground is especially heavy a 6 by 12 in. by 2 ft. filler block is placed under center of cap, and supported by angle braces extending diagonally down on an angle of 45° to posts (Fig. 10). Angle braces are 10 by 12 in. by 3 ft. 6 in. long overall, beveled at each end, and are held in place by standards 6 by 12 in. by 5 ft. nailed to drift face of posts. Timbers 4 by 10 in. by 5 ft. long are generally used for top lagging, while 2 by 8 in. timbers, 3 ft. 10 in. long, are placed "louver"

fashion between posts as side lagging (Fig. 11). Along the sides of each of the drifts at points 25 ft. apart stope chutes are built. They are supported by 6 by 12 in. by 4 ft. chute sills placed horizontally between posts at a height of 4 ft. 11 in. above the sill. One edge of the chute sill is beveled so as to give a dip of 30° to the chutes, which project out into the drift just far enough to be over the side of a car. They are built of 2 by 8 in. timber. While the tramming drifts are being driven a permanent raise, providing the block of ore is thick enough to warrant such an expenditure, is run up in much the same way as described in the "Ray Sub-Level System" to a sub-level which is near the top of the orebody. From this raise on the sub-level a drift is run along one side of the block of ore, and is usually over, and parallel with, a fringe drift on the tramming level. Out of this drift, at intervals of 75 ft., manway or ventilation drifts are driven completely through the area to be stoped. These drifts are placed so as to be over tramming drifts, so that when manway raises are run up from manway chutes, which are built along the tramming drifts near the side of a stope to be, they break into the manway drifts and become, when chain ladders are put into them, openings through which men may pass up and down from one level to the other, and later serve as means of access to stopes from the sub-level.

Stoping.—In starting a stope, raises are run out on flat inclines direct from the stope chutes until they intersect, as was the final practice at Bingham. However, at Ray all drawing-off chutes (stope chutes) for a stope are along the same straight line in the centers of the stopes instead of staggered fashion. After the raises all along a stope have been connected they are widened out to give a stope 15 ft. wide, which is mined upward in the same manner as explained for the sub-level system.

Entries to the stopes from the tramming level are provided by building up a 3 by 3 ft. cribbed manway with the opening into the stope always above and clear of the broken ore. These manways are 75 ft. apart along the side of a stope and lie directly below the manway raises leading to the ventilation or manway level. By placing the cribbed manways directly under the manway raises very little ore falls into them, so they can, at the beginning of a shift, be quickly emptied, allowing men to enter the stopes without loss of time.

Undermining Pillars.—After all the stopes in a block have been finished the work of undermining pillars preparatory for reserve drawing is very simple, as the method is the same as for starting a stope. Pillar chutes, four in all to each pillar in each drift, are built in drift sets midway between each pair of stope chutes and each is so placed as to be directly opposite another. Raises are run out from these pillar chutes, and when all in the same pillar have been connected up, widened out and blasted, the remaining ore in the block, which amounts to 80 per cent. of the total, is ready to be drawn. Winzes, termed dumping chutes, to a

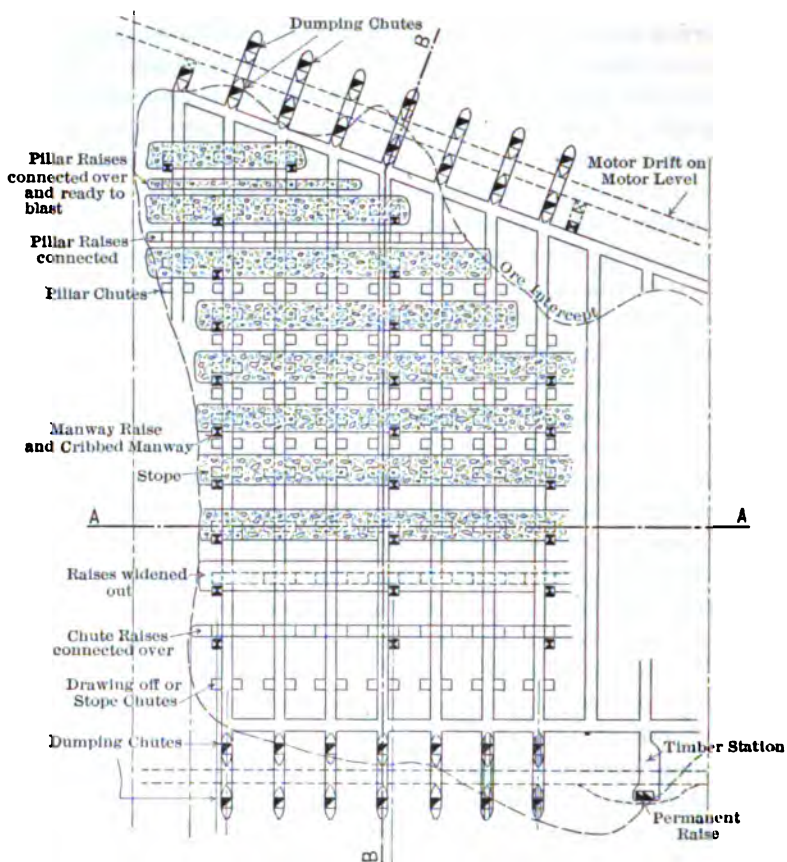


FIG. 9A.—PLAN OF HAND-TRAMMING LEVEL, SHOWING ARRANGEMENT OF STOPE, MANWAYS, STOPE CHUTES, PILLAR CHUTES, AND DUMPING CHUTES.

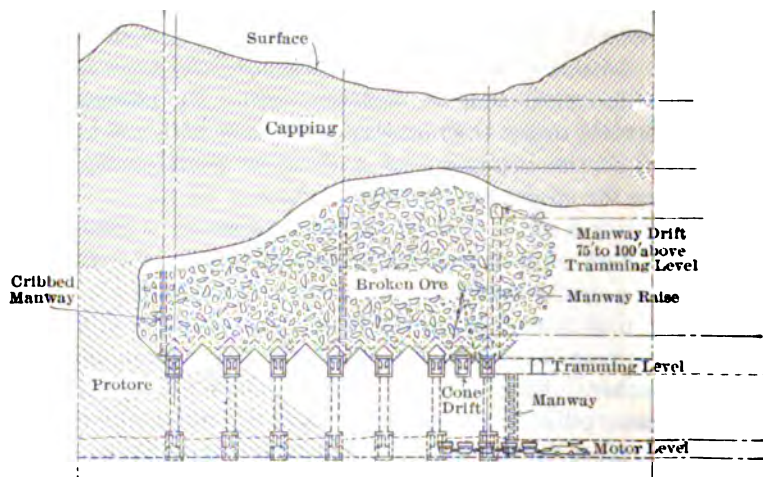


FIG. 9B.—LONGITUDINAL SECTION ALONG A-A, FIG. 9A.

motor level below are provided at the ends of each drift, and after the ore is dumped into these chutes by the trammers it is drawn out into the 5-ton motor cars and taken by electric haulage to the shaft.

Cone Drifting.—In actual practice it has been found impossible to recover all the 80 per cent., as some ore packs in the form of cones over the hogbacks between the tramming drifts; therefore, after all the chutes



FIG. 9C.—METHOD OF UNDERMINING PILLARS.

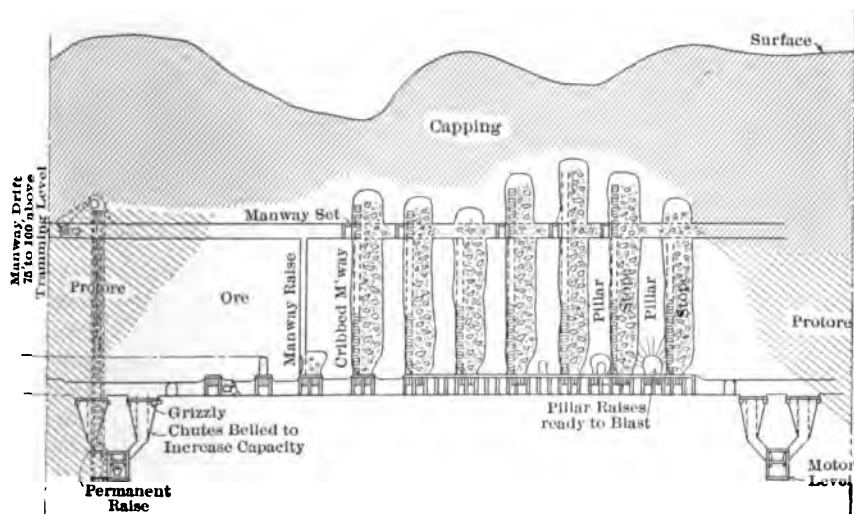


FIG. 9D.—CROSS-SECTION ALONG B-B, FIG. 9A, SHOWING SHRINKAGE STOPES AND PILLARS AND DUMPING CHUTES TO MOTOR LEVEL.

FIGS. 9A TO 9D.—“RAY SYSTEM” OF UNDERGROUND MINING. SHRINKAGE STOPES AND PILLAR CAVING WITH HAND TRAMMING. DRAWING-OFF CHUTES IN CENTER OF STOPE AND TRAMMING DRIFTS AT RIGHT ANGLE TO STOPE.

in the original drifts have been drawn to capping, drifts termed cone drifts are driven down the centers of the 18-ft. pillars. These are timbered the same as the original drifts; chutes are built in every set and drawn, until capping appears. The additional ore remaining in the small pillars on either side of these cone drifts is recovered by gouging out along the sides of the cones.

Reserve Drawing.—In “reserve drawing” a block of ore (that is, extracting the 80 per cent. remaining after all active stoping has been completed) four factors must be kept constantly in mind: 1, recede

toward solid ground; 2, draw rapidly; 3, systematize the drawing; 4, repair broken timbers immediately.

1. **Receding toward Solid Ground.**—It has been found impracticable to draw large blocks of ore evenly over their entire area, owing to crushing of timbers, which necessitates high expenditures for timber repairs. However, in several blocks now completely finished good results were obtained by what is termed locally the "receding method." The two pillars farthest from the permanent raise were the first to be undermined. They, with the stopes adjacent, were the first to be drawn, and in order to have the waste or capping following down over the ore on an incline

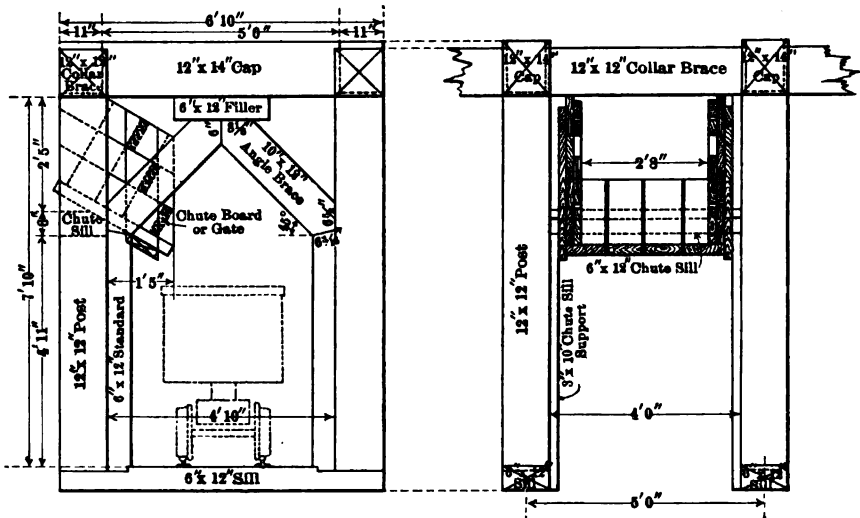


FIG. 10.—DRIFT TIMBERING, HAND-TRAMMING SYSTEM.

dipping away from the main block of stopes the chutes farthest from the permanent raise were drawn harder than those nearer. Not before, but by the time the first row of chutes had been drawn to capping, was the next pillar nearest the two already undermined, blasted, shattered, and made ready for drawing. Proceeding in such a manner no area more than 50 to 75 ft. in width was crushing and caving at any one time, and as fast as the line of chutes farthest away was drawn to capping it was abandoned, allowed to crush, and a new line provided by undermining the pillar next in line. The advantage of this process of receding was that any drift beneath the area being drawn could take weight and crush completely but still be recovered at a very small total cost.

2. **Drawing Rapidly.** It is advisable to pull the ore out quickly once an area has been undermined, since by so doing in most cases all ore can be recovered without any timber repairs being necessary. The determination of the economical proportion of the non-producers, such as

timbermen, timber helpers, chute blasters, screenmen, car checkers, and bosses, in a section, to the actual producers, such as trammers and



FIG. 11.—TRAMMING DRIFT IN HEAVY GROUND. SHOWING CHUTES, ANGLE BRACES, STANDARDS, AND "LOUVER" LAGGING.

muckers, is essential in order to secure the lowest final cost per ton. In other words, as many trammers as can be used without their interfering

with each other should be employed in drawing any block of ground, and therefore it is usually advisable to provide a dumping chute at both ends of every drift. Pursuing this line of reasoning, it might seem advantageous also to have a dumping chute in the center of each drift, but the impracticability of such an arrangement has been demonstrated by the fact that the drifts are thereby weakened and crush, cutting off communication from end to end. Recently in a section with the dumping chutes arranged at the ends of the drifts eight trammers during a shift of 8 hr. produced 770 tons, or an average of 97 tons per trammer. Trammers are given a bonus for each car over a certain number, and are therefore at all times willing to put forth their best efforts.

3. Systematizing the Drawing.—Owing to the great number of chutes being drawn at a time a constant vigilance and record must be kept of each, so that not only the tons remaining, but the assay value, may be known at all times. With such information there is no reason why the total output in tons, assay value, and cost for producing same need to fluctuate from day to day.

The duty of the car checker is to notify the boss at the beginning of each shift, or even the day before, just which chutes are ore and which are waste, so that he may know at all times which should be drawn, where to place his men for obtaining the most beneficial results and just when to undermine the next pillar. It can be seen that by such an arrangement the boss is responsible for the tons produced and the total cost for same, while the car checker must keep the assay value constant; and with such co-operation the best of results must naturally ensue.

4. Repairing Broken Timbers.—If the timbers in any drift crush it is essential to rush the repair work from both ends. A constant supervision of repairs needed must be enforced at all times in order that no ore may be left behind. In most cases drifts taking weight are detected immediately, and the work of drawing out the ore is rushed sufficiently to recover all ore before the drift becomes impassable, thus effecting a big saving for timber repairs. Most drifts crush very gradually, and none collapse suddenly.

HANDLING BROKEN ORE AND WASTE

On each motor level a 3-car tippie (see Fig. 12) placed directly over an ore pocket has been installed. The tipples are similar to those used in coal mining and are arranged to receive three 5-ton motor cars at a time. The 5-ton cars, in trains of 12 cars, are brought to the tippie by motors which weigh 10 tons each. When three cars have been spotted in the tippie by the motorman, the tippie operator turns on the electrical power and the tippie is rotated and the cars of ore turned completely over, allowing the ore to fall into an ore pocket. The pockets are

usually 10 by 28 by 25 ft. deep, holding from 250 to 300 tons. From a pocket the ore is fed through air-dropped gates (Fig. 13) to measuring pockets, size 4 by 4 by 12 ft., holding from 10 to 12 tons—a skip load. From the measuring pocket the ore goes through an air-lift gate to a 10-ton skip, size inside 4 ft. 8 in. by 5 ft. by 10 ft. deep, in which it is hoisted (Fig. 14) vertically to the surface and dumped into an ore bin near the top of headframe. Recently at the No. 1 hoist 364 skips (4,004 tons) of ore were hoisted during a shift of 8 hr. Shafts 1 and 2

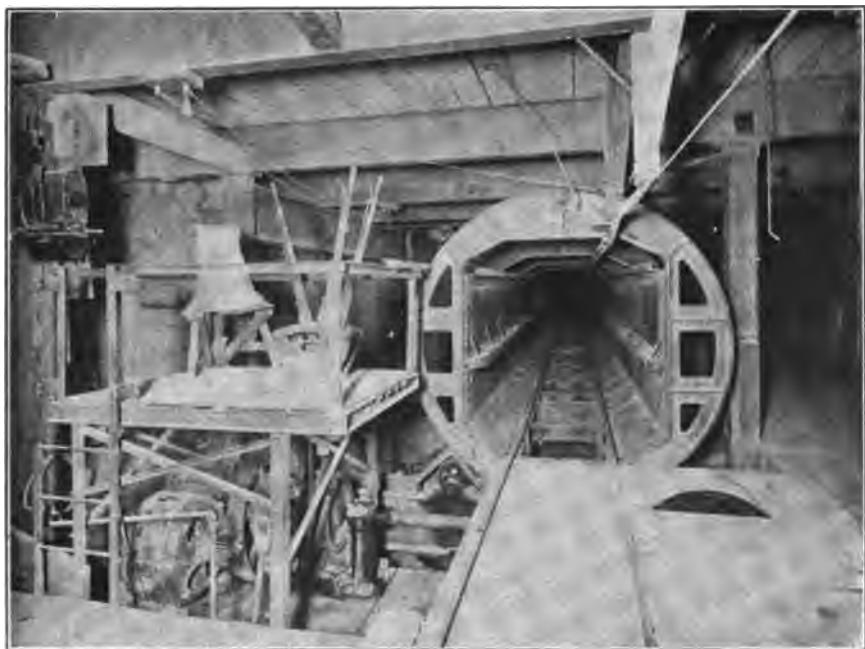


FIG. 12.—TIPPLE, WITH ORE POCKET DIRECTLY BELOW IT.

have two compartments, size 6 by 6 ft. 10 in. The ore then passes through one of four air-lift gates and over a grizzly. The oversize goes to two size 8, style K Gates gyratory crushers, and unites with the undersize from the grizzly, then into a bucket elevator which hoists the ore to the top of building and allows it to pass over an inclined screen chute to two Garfield rolls, size 72 by 20 in. The undersize from the screened chute unites with the crushed ore from the rolls and is fed on to an inclined conveyor belt which takes the ore to the top of a storage bin. (At No. 2 shaft the storage bin holds 5,000 tons while the one at the No. 1 shaft holds 25,000 tons.) From these bins the ore is drawn off through basket gates into railroad cars of 60-ton capacity and shipped to the

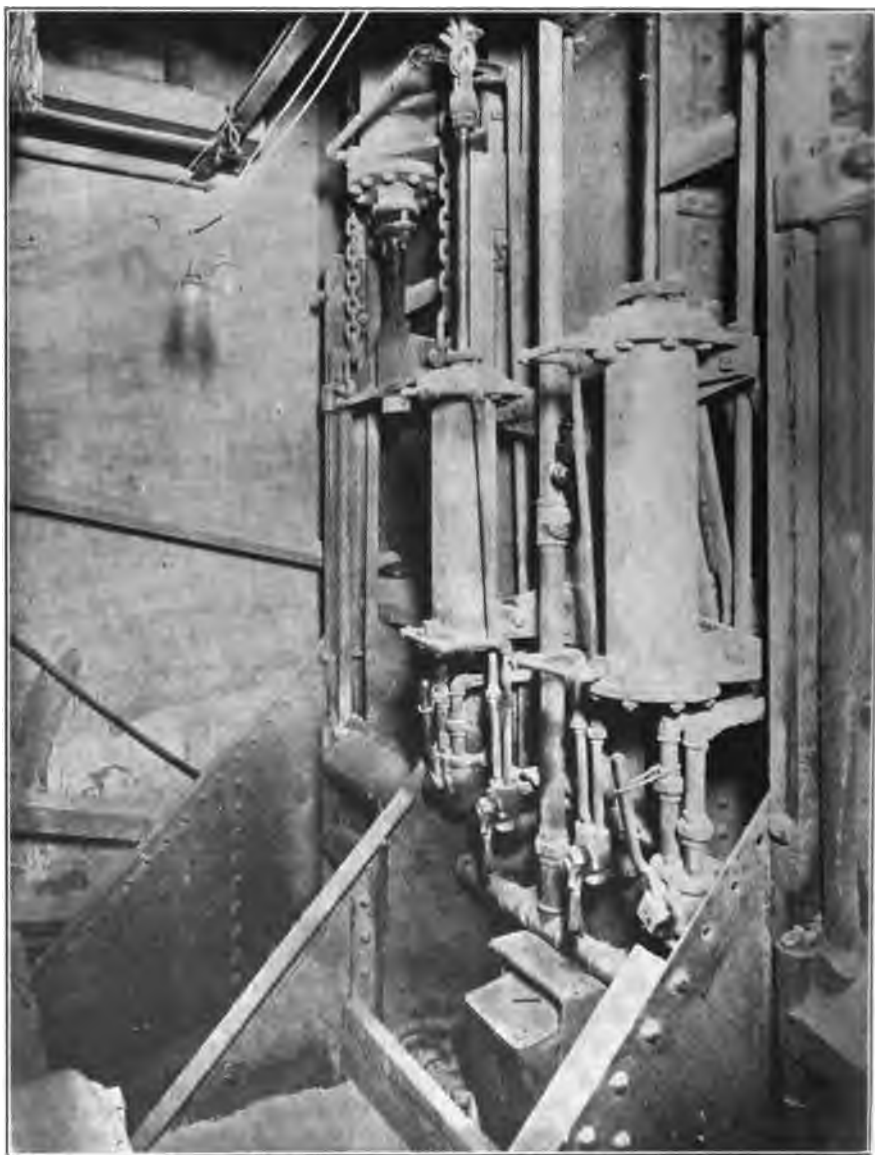


FIG. 13.—AIR-DROPPED GATES, BETWEEN ORE POCKET AND MEASURING POCKET.

concentrator at Hayden, 23 miles from Ray. The ore is usually sent to Hayden in trains of 38 cars, or about 2,400 tons to a trip.

The waste encountered during development and prospecting work is loaded into V-shaped side-dump cars of 45 cu. ft. ($2\frac{1}{4}$ tons) capacity. These cars are hauled by the motors to a winze, where cars are dumped, the waste falling to a pocket directly over an inclined shaft. From the pocket the waste is fed through hand-operated arc gates to skips, size 60 cu. ft. (hoisting capacity $2\frac{1}{2}$ tons). The skips are hoisted to the



FIG. 14.—THE ELECTRIC HOIST.

top of a 30° incline shaft, where the waste is dumped into a cylindrical iron bin of 10,000 cu. ft. (500 tons) capacity. The No. 1 incline shaft (material hoist) has three compartments, each 5 by 7 ft. Two compartments are used for supplies and waste, while the third contains air pipes, water pipes, electric wires, and a stairway for the men. No men are hoisted or lowered, so that all those working on the lower levels use the stairways (Fig. 15).

The waste is drawn out of the bin into small crab cars and distributed for ballast on railroad, or filling (gobbing) square sets at No. 3 mine.

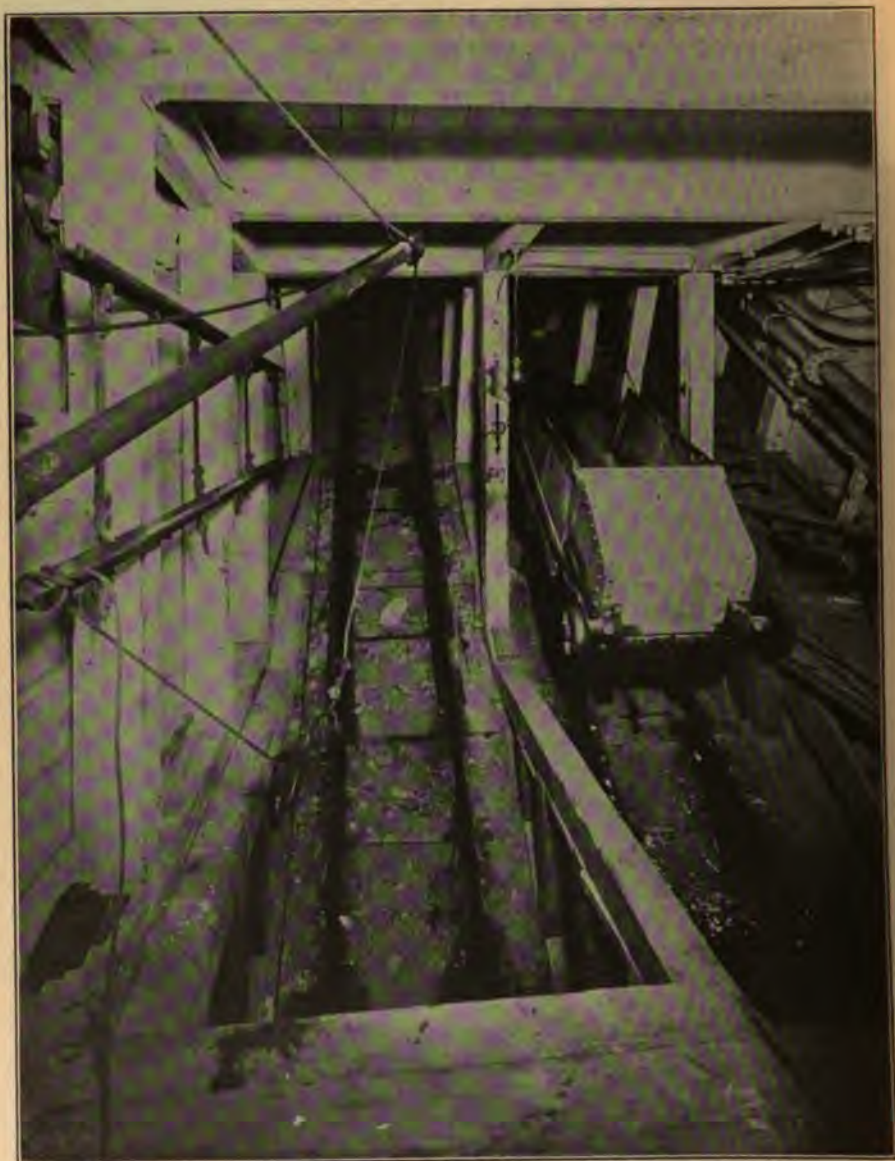


FIG. 15.—INCLINE OR MATERIAL SHAFT. SHOWING SKIP, TWO HOISTING COMPARTMENTS, AND STAIRWAY; ALSO AIR AND WATER PIPES, AND ELECTRIC WIRES.

ENGINEERING WORK IN CONNECTION WITH THE "RAY SYSTEM"

In efficiently handling the "Ray systems," engineering is of necessity the paramount fundamental. From the time churn drilling is started to the final drawing of a stoped area, the engineering department, in co-operation with the management, must be wary and alert.



FIG. 16.—CHUTE BOARD. SHOWING THE PADS WITH NUMBERS FOR DETERMINING THE ORE REMAINING IN CHUTES.

Records, maps, and drawings are kept by engineers of each successive step leading up to and during active stoping and reserve drawing of every block of stopes.

A transitman with his helpers is responsible for correct reports on grades, bearings, timbering and proper connecting up of all drifts, inclines, raises, shafts, etc., and plotting of same on maps and sections.

The sampling department takes groove samples at intervals of 5 ft. in all headings, and records assays on maps, sections, and in permanent office record.

"Stope engineering" covers the work of efficiency in underground mining and in making reports and suggestions from time to time for betterment in operation; keeping records of the tonnage obtained and cost of producing ore from each stope, and an aggregate for each block of stopes; making semi-monthly cost and tonnage reports, which are posted on a bulletin board at "Candle House," where all the bosses may see and know the results that each is obtaining in his particular section of the mine; taking samples from time to time along the back of each stope while active so as to keep bosses posted as to whether the stope back is in ore or waste (these assays are entered upon the stope sections); also calculating tonnage and assay value of each block of ore with a proportional distribution of same to each chute to be drawn, therefore by keeping account of all ore drawn from each chute, the amount still remaining is known, and an even settlement of the ore and capping can be effected; keeping the management informed not only as to the surface movement over each area, but also as to the ore remaining in each and every chute in the various sections of the mines by means of "chute boards." These boards are 6 ft. wide, $7\frac{1}{2}$ ft. high and 1 in. thick, with holes 2 in. apart horizontally and $1\frac{1}{4}$ in. apart vertically (see Fig. 16). Each hole represents a chute, and is drilled large enough to receive a bolt $\frac{3}{16}$ in. in diameter. Bolts are $3\frac{1}{2}$ in. long to receive pads of thin paper or tickets which are printed with consecutive numbers. Each ticket is 1 by $1\frac{3}{4}$ in. in size and perforated through the center; the top half contains a hole for the bolt, and the lower half, a number. When ore is drawn from any chute, tickets indicating the amount of ore drawn are torn from the pad representing that chute. A ticket showing the total amount of ore in each chute is placed over the upper half of the pad, so that it is easy to determine the tons of ore still remaining by deducting the number on the lower half from the one on the upper half of the pad representing that particular chute. Figures are placed across the top of a board to designate the drifts, while others down the side indicate the chutes.

Permanent Stope Records are kept by the stope engineers showing the tons produced, together with cost and assay value of same, for each stope, both during active stoping and reserve drawing; also for pillar drifting, cone drifting, and cone stoping. The information for these records is obtained as follows:

A stope card (Fig. 17) is turned in to the engineering office by the bosses after every shift for each class of work. Each card shows the labor and material used during that shift chargeable to stoping operations. The items chargeable to stoping operations in the sub-level system are:

1. All work of every description above the first sub-level within the block of ground to be stoped, preparatory for stoping, except that pertaining to manway drifts and stope drifts.
2. Belling and widening out of chute raises.
3. After all active stoping has been completed and reserve drawing begun all work of every description within the stoped area on and above the motor-haulage level.
4. All work pertaining to pillar drifting, cone drifting, and cone stoping.
5. All work of recovering timber after reserve drawing has been finished and section abandoned.

FORM M 21-REV

RAY CONSOLIDATED COPPER CO.

MINE NO. _____ LEVEL _____ STOPE REPORT _____ 191 _____			SHIFT _____					
PLACE OF WORK			SHIFT					
LABOR			MATERIAL				PRODUCTION	
	CONTRACT	COMPANY					CHUTE	CARS
			POWDER STICKS					
LOADERS			CAPS FUSE					
MUCKERS			CANDLES LAMPS					
TRAMMERS			LUMBER	AMOUNT	SIZE	LENGTH		
			Pests					
			"					
CHUTE BLASTERS			Caps					
			"					
SCREENMEN			Angle Braces					
			Collar Braces					
STOPMEN			Bills					
			Lagging					
			"					
			Standards					
TIMBERS	American		Butt-A					
	Medium		Chutes					
			Ladders					
	American		Heavy Timber					
	Medium		Other Timber					
SAFETY--The co-operation of all employees in our efforts to prevent accidents is earnestly requested.							Signed _____	STOPE NO. _____

FIG. 17.—STOPE REPORT BLANK.

In the hand-tramming system the charges to stoping operations are:

1. All work of every description above the tramming drifts within the area to be stoped, except manway drifts.
2. After active stoping has been finished and reserve drawing started, all work on the tramming level and sub-level within the area stoped.
3. Cone drifting with the necessary timbering; chute building and cone stoping in same.
4. Recovering all good timber after cones have been drawn and section abandoned.

A car report (Fig. 18) is obtained from the car checkers showing the number of cars drawn from each chute, with a grand total for the section.

Form 35-34

CLASS _____

PAGE _____

MINE No. _____

LEVEL No._____

SECTION

DATE _____ 191_____

SHIFT_____

TIPPLE _____ **CARS**

GRAND TOTAL_____CARS

FOUND LOADED_____CARS

LOADERS

CHUTE BLASTERS_____

SIGNED

to the trammers, who are responsible for their proper delivery to the assay office. At noon of the following day an assay sheet is posted in the

engineering office where the car checkers may see the value of the samples taken the previous day and regulate drawing accordingly.

Stope Record Sheets

In the monthly record a separate account is kept for each stope. Each account consists of two pages. One page is used for labor and material, forming the cost sheet (Fig. 19). The other sheet is used for a carirecord and forms the tonnage sheet. All cost sheets are kept together, likewise the tonnage sheets. Each sheet is ruled to accommodate one month's run. Several other extra sheets are also kept as a summary check on the entries of the others.

All entries on the cost sheets are made in terms of shifts or fractions of a shift under the various labor headings. Material is debited under the proper heading according to the units reported for each class. A price list and wage scale are kept near at hand. At the end of each 15-day period the total is found for each item and multiplied by the price. Cross additions are then made to get total labor and total material. The sums of these give the total labor and material.

The car reports from the various stopes are then entered on the proper tonnage sheet in terms of cars. The total in the right-hand column gives the total cars per shift. To the right of this column the corresponding assay is entered and this multiplied by the total cars per shift gives the cars per cent. per shift, which is entered to the left of the total car column. At the end of each period the vertical columns for each chute are added and their sums must check with the sum of total cars per shift column. Each of these totals is then multiplied by a factor to reduce each from cars to tons. The cars-per cent. column is then footed up and multiplied by the same factor to get tons-per cent. Tons-per cent. is then divided by the total tons to get the proper assay average for the period. Similarly, the total cost of labor and material is divided by total tons on each sheet to get the cost per ton.

As a check on the price multiplication the shift sums from the bottom of each column are entered by stopes on a separate sheet and these correspondingly totaled up. Each sum is then multiplied by its proper rate and the total of this should check the total of the labor and material columns of all the stope sheets. Any error is thus quickly traced.

As a check on the car entries a separate sheet is kept for the total cars per shift as given on each car report. The sum of these totals should check with the sum of the total cars per shift column of all the stope tonnage sheets. The tons-per cent. entries have no independent check, but all assays are carefully entered and all multiplications for cars-per cent. checked by casting out the nines. Average assays and costs per ton are obtained by the slide rule with sufficient accuracy, but are checked

RAY CONSOLIDATED COPPER CO.

STOPE

MINE NO.

RAY, ARIZONA

STOPE COST RECORD

LEVEL.....191

Date	Shift	LABOR													MATERIAL										TO DATE			
		Loaders	Trammers	Chute Blasters	Screen Men	Stope Boss	Stopemen	Pickmen	Timberman	Timber Helpers	Car Checkers	Assaying	Others	Total Labor	Powder	Caps	Ruse	Candles	Machine Drill Exp.	Timber	Total Ma- terial	Total Labor & Material	Production, Tons	Cost per Ton	Total Labor & Material	Production, Tons	Cost per Ton	Date
1	1 Day																											1
	Night																											
	to Day																											
	Night																											
	15 Day																											15
	Night																											
	Total for Period																											
	Amount																											
	15 Day																											15
	Night																											
	to Day																											
	Night																											
	31 Day																											31
	Night																											
	Total for Period																											
	Amount																											
	Total for Month																											
	Amount																											

Fig. 19.—Stope Record Sheet.

by repeating. So far the checking has only been for the benefit of the office work. As a check on the stope cards themselves, or any missing cards, the various costs for the period are grouped under the headings of: 1. Breaking; 2, tramming; 3, timbering; 4, engineering, etc.

The first includes stope bosses, stopemen, and pickmen.

The second includes loaders, trammers and muckers, chute blasters, and screenmen.

The third includes timbermen and timber-helpers.

The fourth includes car checkers, assaying, engineers, etc.

The sum of these items should equal the total labor as found from all the cost sheets. To this figure is added total material. The final sum must check with the total labor and material from all cost sheets. The totals thus segregated are then comparable with the cost accounts as kept by the clerical force, these being independently made up from the mine timekeepers' reports.

To reduce and concentrate the bulk of these monthly records, a separate cover is used for the summary sheets. The form for these sheets is identical with those used for daily entry. The totals for each column in terms of cost only are entered at the end of each period with the date. A separate sheet is kept for each stope. The upper half of each sheet is used for costs and the lower half for tonnage. On this latter the entries are made in terms of tons and tons-per cent. only. On the extreme right of each sheet are carried the totals to date of labor and material, production in tons, and tons-per cent. The average costs per ton and assay to date are also calculated and entered.

COMPARATIVE ADVANTAGES OF THE TWO SYSTEMS NOW IN USE AT RAY

It must be remembered that the sub-level system is only applicable in sections of a mine in which the orebody covers a considerable area and has a height exceeding 100 ft. The development of the stope laterals is necessarily expensive; the timbering is costly; and considerable expense is incurred in equipping and maintaining the drifts for motor haulage. Unless the orebody is of sufficient area and height to repay the cost of the initial large expenditure, the sub-level system should be abandoned in favor of the hand-tramming system. The great advantage of the motor-haulage system is that it is capable of producing a comparatively large tonnage, not only in the operation of putting up the active stopes in the section, but more especially when reserve-drawing operations are started. The tonnage per shift that can be produced from a reserve in a sub-level section is practically only limited by the efficiency of the electric-haulage system and the capacity of the shaft and pocket equipment. If such conditions are at hand, the sub-level system will produce a large output at a very low cost. However, in sections of the mine where

the ore is of less extent and of a height less than 100 ft., the hand-tramming system, primarily owing to its less initial expenditure, will be found more economical to install (Fig. 20). Its advantages over the sub-level system may be enumerated as follows:

1. Less development work is required, with a consequent lower total expenditure; only a small capital outlay is necessary before stoping can be started, and a big tonnage supplied; no great amount of money is tied up for initial expenditures, whereas with the sub-level system all the big expensive motor-haulage drifts must be completed before any ore is obtained from stoping; also the outlay for track, pipe, and trolley wire is high.

2. The total outlay for timber is less, hence the system is adaptable to localities where timber is expensive and difficult to obtain.

3. The total cost for maintaining drifts is lower. When one of the small tramming drifts caves it can readily be repaired with a small outlay, whereas if any of the motor drifts in the sub-level system crushes, the repair work is not only expensive, but the time required may necessitate cutting down the daily output.

4. A higher final extraction can be obtained, as chutes can be built in every set in both the original and cone drifts, thus supplying drawing-off chutes at frequent intervals at the base of the block of ore; but with a sub-level only 30 ft. above the haulage level it is not advisable to cut up the pillar below the sub-level with drawing-off chutes too close together, owing to the danger to the important haulage drifts below; also, the work of running up the raises to the sub-level increases the total cost per ton of ore obtained from a block.

5. The hand-tramming system is adaptable to shallow as well as deep orebodies because of the small outlay for initial development work.

6. The rapidity with which a block can be opened up and made ready for producing ore is an additional advantage of hand tramming.

CONCLUSION

The "Ray systems," although as yet comparatively new in their application, have amply justified all expectations.

The systems have been used in both hard and soft ground. In blocks where the ground is very hard the stopes are carried wider than the regulation 15 ft., thus leaving the remaining pillars narrower and more readily shattered by undercutting. This feature of narrowing and widening stopes according to the nature of the ground insures at all times the safety of the miners, and accounts for the fact that very few accidents occur actually within the stopes. The stopes are under constant supervision, with the backs closely watched and inspected before each drilling.

The method of mining is adapted to the use of stoper (air-hammer)

machines, so that only a few men are needed in producing a big tonnage. All stopes are mined in much the same way and systematically, consequently the men soon become proficient in their work. Since all blasting is done at the end of each shift, but little time is lost. Practically no timber is used above the motor or tramming levels, and in comparison with other caving systems little raising and drifting is done. With the sub-level system even the "stope drifts" on the first sub-level may be done away with by simply connecting over raises direct from chutes on the motor level, as outlined for "Undermining Pillars."

In conclusion, it is but necessary to emphasize the great flexibility of the systems. Practically any number of men can be used during active operations and a corresponding tonnage produced, and when circumstances require it the number of men and tonnage can be either increased or decreased without materially affecting the total direct mining cost per ton.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 129 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Sliding Royalties for Oil and Gas Wells

BY ROSWELL H. JOHNSON,* PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

THE principle of sliding or graduation in royalties is accomplished either by the block, period, cumulative, or class method.

The block method calls for a very low royalty rate on all oil produced up to a certain amount. This amount would normally be that which is believed to suffice to repay the cost of the well. A higher rate would be charged after the well is making a profit. This method, common in many industries, is peculiarly inadaptable to any industry such as this where the productivity per day of the units gradually and inevitably decreases. This, for reasons given later, is because it forces an early abandonment of the well and so offends against conservation.

In the period method, the royalty rate is changed by some definite amount when the well produces less than a specified quantity per day or other given unit of time. While simple to explain, it operates badly because a great deal is made to depend upon a small reduction in production, whereas the reduction as a matter of fact declines rather erratically owing to the exigency of lease management, connecting of tankage, and vagaries of the gaugers.

These disadvantages are obviated by the cumulative method. In this method, all the production less than a certain amount per week has one rate of royalty, or none at all, and the oil produced in excess of that amount pays an additional rate of royalty. In the long run the calculation is probably no greater than with the period method, for simple tables will be published and printed in the oil papers that can be cut out, enabling quick readings to be made. Even without the table the calculation is easy with the few royalty rates, two or three, which would be used. In this method, the amount paid in royalty declines gradually, rather than suddenly as in the period method.

The following reasons might be advanced for the adoption of cumulative royalties:

1. To distribute the burden in accordance with the capacity to pay. This principle is a unique one in business, although familiar in taxation. This is because it does not distribute reward in proportion to efficiency.

To fail in this is to allow an industry to degenerate in its progressive and alert attack on the technical and economic problems of the business. It is not then to produce a more equalitarian and less just distribution of wealth among producers themselves, that the cumulative royalty is here urged.

2. The fear has been expressed that if sliding royalties are adopted, too steep a graduation might be used, which would have the effect of destroying the occasional high rewards essential to the producer to recoup him for the heavy expenses of the inevitable proportion of dry holes. Such a result would necessitate a great increase in the cost of oil to the consumer to produce higher profits on small wells in order to cause the producer to continue his activities. Such a result would be a social loss, but the fear in my opinion is not justified, for the land owner would not get such steeply sliding royalties without serious sacrifice in the bonus, which he would usually prefer not to make.

3. Another advantage of the cumulative royalty is to prevent the excessive flat royalties now frequently offered for promising land, such as the 50 per cent. on the Cimarron River bottoms and the 25 per cent. that we occasionally hear of in other fields. These royalties always lead in a few years to an unpleasant threatening and bargaining between land owner and lessee resulting in successive agreements to reduce the royalty rate. This awkward process, it is true, does accomplish a gradual reduction of the royalty rate. But the asperities of such negotiations are very annoying and sometimes lead to a premature abandonment of the well with a consequent serious offense against wise conservation. The cumulative royalty, while avoiding the difficulties just referred to, still retains a flexibility by virtue of which the producer who desires to may increase the royalty rate, when the well can stand it, in this simple way transferring some of the speculative profits to the land owner in lieu of bonus. This bonus the producer may not be able to pay, or the land owner may prefer not to accept.

4. The fourth reason is the most weighty. The cumulative royalty lengthens the life of the well and increases the percentage of the oil which is recovered. If the royalty is one-sixth and the maintenance and interest on the "junk" is $83\frac{1}{2}\%$ per day, then a well must be abandoned when its net income to the producer reaches that amount. Yet the gross income is still \$1 per day, and if the decline of the well is one-sixth in a year (a common decline in old wells in Oklahoma) it might continue to produce for a year longer except for the royalty. Thus, 300 bbl. per well might easily be saved by a mere royalty adjustment. If the Osage Nation is leased at one-sixth, as now threatens, all its wells will be prematurely abandoned and a most serious loss result.

A fourth method using a sliding rate is the class method. Here the wells are classified at the beginning and a different rate used with each

class. The classification is based upon the ratio of value of product to the cost of production. Normally some one variable factor in either of the items above would be the basis of the sliding. The one which will be most used and which I recommend is the depth of the well.

To charge the same royalty rate for wells of 600 ft. as for those of 3,000 ft. has the effect of promoting "post-hole" drilling, as we call holes of inadequate depth. Such a wide distribution of shallow dry holes, where the untouched underlying strata are worthy of test, results not only in a direct waste, because the area must later be redrilled, but also in an indirect waste, because later operators, being in doubt as to the depth of the older holes, are afraid to risk drilling in territory thus improperly condemned. The classes should be few, and all wells in one pool should be in one or another of the classes.

For instance, if one-sixth royalty be charged in the Osage Nation then the deeper sands of the western Osage would not be as systematically or economically prospected, as would be the case if the royalty was graded by depth. I suggest one-sixth for oil in pools averaging above 2,000 ft.; one-eighth from 2,000 to 3,000 and one-twelfth below 3,000, for the present leases with their partly developed production. For the rest of the Osage Nation I would suggest one-eighth above 2,800 ft. and one-twelfth below it.

Let it be remembered that the Mississippi lime (Boone chert) lies deeper than 3,000 ft. in much of the western Osage and that no test is thorough unless the drilling is continued until that formation is reached, when the upper horizons prove barren. Let no one suppose that because the Cushing and Boston pools have been great producers all the deep pools of the Osage will be of that kind. West Virginia has plenty of deep small producers, and so has the Bartlesville sand. The system above outlined would encourage thorough prospecting.

The objection that one-twelfth and one-sixth are too great a difference will arise in the minds of those who think only of successful wells. It seems none too great when one remembers the amount of futile drilling which must be paid for by proceeds from those that are successful. To consider next the requirements of a proper system of sliding royalties:

1. It must be simple, so that it can be readily understood.
2. It must not be expensive or difficult to calculate.

It follows from these two considerations that it should have but few rates.

3. It must avoid uncertainties as to the time or point of changing rates.

4. Most important, it should permit the well to be pumped until its gross income has fallen to its maintenance cost.

To accomplish this last, I suggest an exemption from royalty on the oil equal to maintenance and interest on the junk. The slight loss to the

land owner will be met ordinarily in the bonus, but sometimes by a higher royalty. For practical reasons the amount that should be exempted, instead of being exactly equal to maintenance and interest on the junk, would be fixed for the field at an approximate integral number of barrels, ordinarily one. When several wells discharge into one field tank, they may be averaged.

5. It should not encourage post-hole drilling as opposed to thorough prospecting.

6. It should bear some relation to the cost of production.

For these two reasons we should in addition classify the royalty according to depth.

The objection which might be raised that the land owner would in some cases receive nothing for the use of the land during the last months of the history of the well is met by the fact that he has received advance payment for such use either in bonus or in large royalty during the early history of the well. It would be absurd to say that it would be illegal to pay a lecturer a royalty only on the tickets sold after other costs have been met.

Gas Royalties

In some fields gas may be so cheap and wells so isolated that it is still infeasible to meter from the lease. But where a gas royalty is paid or indicated as the better method, the general principles laid down above apply in even a greater degree than to oil.

This is because a gas well is so frequently abandoned because its volume is not enough to pay the fixed rental for a well; also because it is the small wells about to be abandoned which have especially heavy maintenance by reason of the necessity sometimes of pumping water, or more often of installing pumping stations.

To meet this situation I recommend for gas wells a stated royalty, with the exemption of a certain amount per week, roughly estimated in advance, equivalent to cost of maintenance and interest on junk. This exemption is to be increased to a higher proportionate exemption if, and when, water pumps are installed at the well or pumping plants are required to raise the pressure sufficiently to put the product into the main lines.

We conclude then that the flat royalty offends against a wise and economic exploitation of our oil and gas deposits.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Concentrator of the Timber Butte Milling Co., Butte, Mont.

BY THEODORE SIMONS,* E. M., C. E., BUTTE, MONT.

(San Francisco Meeting, September, 1915)

I. INTRODUCTION

PERMISSION to present this paper at the February, 1915, meeting of the Montana Section of the American Institute of Mining Engineers was liberally granted by W. A. Clark, Jr., President and General Manager of the Timber Butte Milling Co., and by the representative of the Minerals Separation Co.

For the many courtesies shown the writer on his visits to the mill and for information supplied, he is indebted to all the officers of these companies, especially to W. N. Rossberg, W. D. Mangam, Hamilton Cooke, Jr., T. M. Owen, T. C. Wilson, R. McGillivray, L. L. Quigley, C. Bartzen and others. The writer takes this opportunity to herewith express to them his appreciation and sincere thanks.

Location of Mine and Mill

The concentrator of the Timber Butte Milling Co. was built to treat the ore mined at the Elm Orlu mine of the Elm Orlu Mining Co., which carries zinc and lead in sphalerite and galena; copper in bornite, chalcocite, tennantite, and tetrahedrite; and gold and silver either free or chemically combined with the sulphides of the base metals. The gangue is chiefly quartz, barite, fluorite, rhodonite, and rhodocrosite. The iron in the ore occurs as iron pyrites.

The Elm Orlu mine is situated in Butte, Silver Bow County, Mont., near the northern border of what is locally known as the Butte sulphide-copper belt, about one mile north of the Anaconda mine, and adjoining the Black Rock zinc mine of the Butte & Superior Co. on the southwest. The mill is on the northern slope of Timber Butte, a prominent topographical landmark in the Butte district. This site permits to a large extent the gravity system of ore flow; it presents a convenient and extensive area for a tailings pond; and it is comparatively near to the source of water supply and to existing railroads.

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The mill is 4.5 miles from the mine in an air line, or 13 miles by rail. The ore is loaded on railroad cars at the mine, taken over the B., A. & R. Ry. high line to Rocker, and thence by the Chicago, Milwaukee & St. Paul Ry. to the Timber Butte exchange, a siding on the Milwaukee

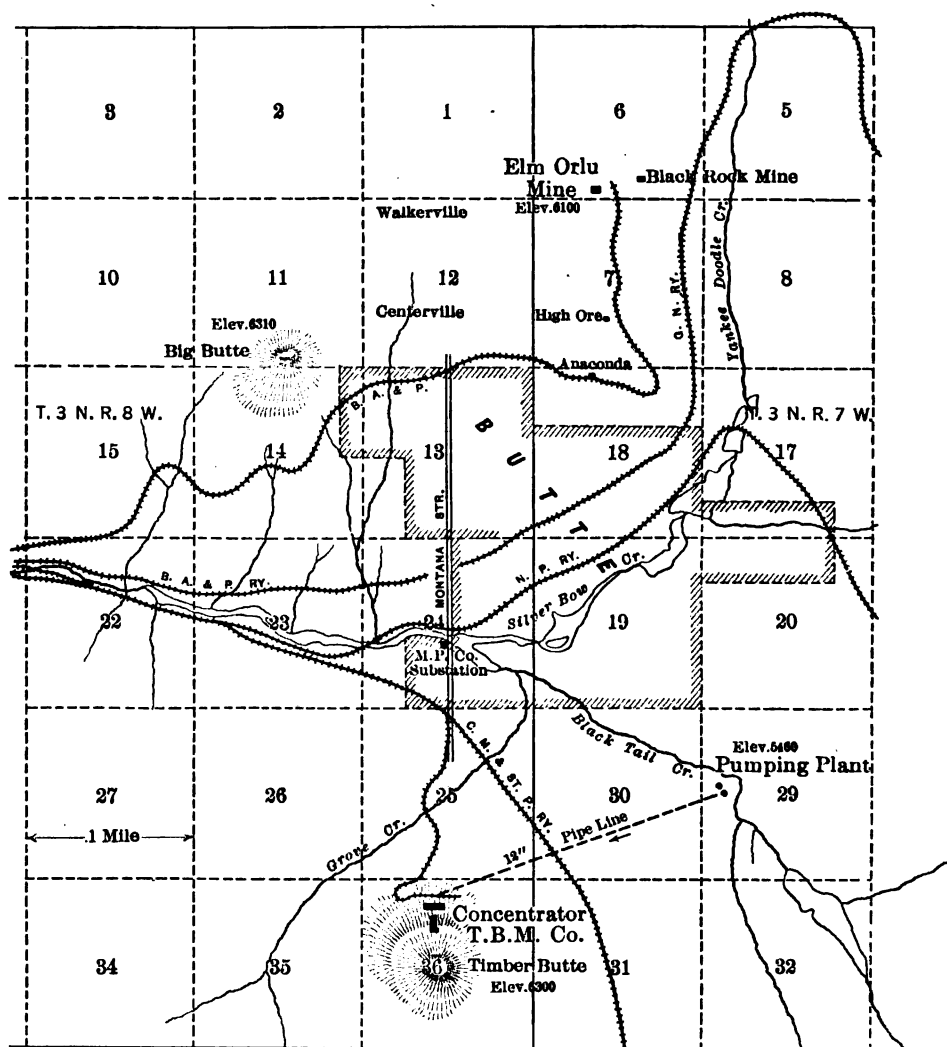


FIG. 1.—MAP OF BUTTE AND SURROUNDING TERRITORY, SHOWING PLANT OF TIMBER BUTTE MILLING CO., AND ORE AND WATER SUPPLY.

line where it crosses Montana Street, a distance of 11 miles. From this siding the cars are hauled, two at a time, to the mill by an electric 400-hp. Baldwin locomotive, a distance of 2 miles. The concentrates are taken from the mill to the sidings on the Milwaukee or the N. Pacific

line where the respective railroad companies receive them for transportation to the smelters.

Fig. 1 shows the location of mine, mill, railroads, and pumping plant.

Design of the Concentrator

The treatment of the Elm Orlu ores presented the usual problem of making a series of products the combined market value of which, after deducting mining and milling costs, would leave the maximum of profit.

The solution of this problem calls for two essential features in a mill: simplicity in the processes employed and economy in operation. Both these features are conspicuous in the plant of the Timber Butte Milling Co., the design of which was based on the results of extensive tests and experiments, covering a period of seven months. In the course of these tests, carried on by a trained staff under the direction of W. N. Rossberg, the company's chief metallurgist, various processes and machines were tried out, the ultimate aim being to make at the least expense the following marketable products:

1. A product for the copper smelter, containing, as nearly as possible all the copper, with some iron and a minimum amount of lead, zinc, and insolubles.

2. A product for the lead smelter, containing, as nearly as possible, all the lead, with some iron and a minimum amount of copper, zinc, and insolubles.

3. A product for the zinc smelter, containing, as nearly as possible, all the zinc, with a minimum amount of copper, lead, iron, and insolubles.

The gold and silver contents of the original ore were permitted to go with each of the above products in varying amounts, as shown in Table I.

During these tests, treatment costs and percentages of extraction were carefully estimated and balanced against each other. The result is the present design and flow sheet of the plant, which by a judicious process of elimination was reduced to two essential operations: table concentration, followed by flotation, the latter in accordance with the Minerals Separation Co.'s patents.

In order to arrive at the most suitable or profitable flow sheet for the Timber Butte mill, the peculiar composition of the Elm Orlu ore had to be taken into consideration. In the first place, there is not sufficient lead in this ore (0.75 per cent.) to warrant a large expense for its recovery. It is, moreover, so intimately combined with the zinc and other sulphide minerals, down to the finer sizes, that it is difficult to make a separation. The percentage of the iron in the ore (3.29 per cent.) is likewise hardly high enough in proportion to the zinc (18.52 per cent.) to permit a satisfactory separation by gravity concentration; yet it is sufficiently high to

reduce the grade of zinc concentrate very materially, if not removed. A large proportion of the iron is combined with the zinc and copper, which reduces the possibility of separating the iron from the other metals, and limits the expenditure which may be allowed for its removal. It therefore becomes necessary to provide at moderate cost a lead-iron product rich enough for re-treatment. This is accomplished on Wilfley and James roughing tables.

These tables also make a finished zinc concentrate which has averaged 3.73 per cent. of insolubles for the six months up to Dec. 1, 1914. This combined with the fact that by flotation and final separation the insolubles in the concentrates are reduced to 5, 4, or even 3 per cent. eliminates any possibility for jigging operations, at least for coarse sizes, since it is difficult to pick out from the ore any zinc particles of, say, $\frac{1}{2}$ -in. size, containing less than 10 or 12 per cent. of insolubles. This is due to the presence of small bands and crystals of gangue material, not visible to the naked eye. Even at $2\frac{1}{2}$ -mm. size, few particles containing zinc, contain less than 8 or 10 per cent. of insolubles. Jigging is also objectionable on account of the resultant high percentage of insolubles in the zinc concentrate, the large consumption of water, and the large floor space required by jigs having a given capacity; all of which features add heavily to first cost of plant and to operating costs.

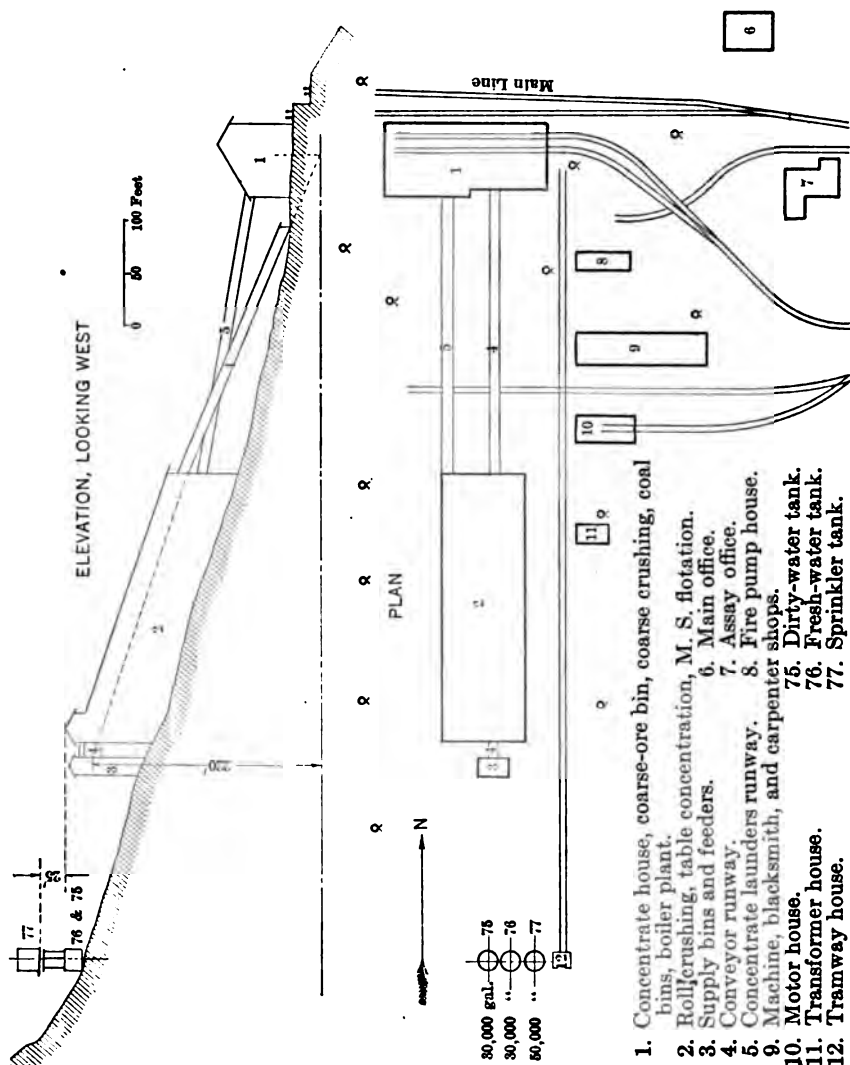
The recovery of 25 or 30 per cent. of the total original zinc in the ore on the roughing tables, is satisfactory from any point of view. The cost is small, the water consumption low, and the capacity large. The economy of this operation becomes still more apparent when it is considered that on these tables the lead and iron is enriched to a point where re-treatment and the separation of the lead and iron from the zinc become economically possible.

Were it not for the fact that by the skill of the company's engineers the efficiency and economy of the roughing operations were brought to a very high state of perfection, it would unquestionably have been more profitable to eliminate all gravity concentration in favor of a simple flotation plant. For it was found that with the exception of the roughing operations, no other methods of gravity concentration could compete with the flotation process in the cost of producing a certain grade of zinc concentrates.

No commercial method, other than smelting, has yet been found for an efficient separation of the copper from the zinc. The company's engineers, however, are making experiments with a view to accomplishing a satisfactory separation of both the copper and silver from the zinc.

The mill proper consists of three separate buildings, connected by runways for conveyors and concentrate launders. The lower building, 155 by 70 ft. in size, contains the concentrate plant and bins, the coarse-ore bins and coarse breakers, the boiler plant and coal bins. The next

higher building, 260 by 80 ft., contains the roll-crushing, table-concentration, and flotation plants. The highest building, 20 by 30 ft., contains the supply bin and feeders. All three buildings are frame structures on concrete foundations with concrete floors. The main office (36 by 45 ft.); the assay office (42 by 52 ft.); a 127 by 30 ft. build-



ing, containing machine, blacksmith and carpenter shop, and the 57 by 25 ft. motor house are frame structures on concrete foundation. The 18 by 50 ft. building for the fire pump and the 20 by 39 ft. transformer building are of concrete throughout.

An inclined surface tramway, 800 ft. long, runs from the bottom to the top of the mill, along the east side of the buildings, delivering supplies on any level. It is operated by a 37-hp. electric hoist and is good for a 10-ton load.

Fig. 2 shows the location of buildings, tanks, fire hydrants, tramway and railroad tracks.

The plant will be described under the following titles: Flow Sheet; Operation Details; Power; Water Supply; and Fire Protection.

II. FLOW SHEET

(See Fig. 3)

1. Coarse Crushing

Upon arrival at the top of the lower mill building, the ore is dumped from the railroad cars into two crude-ore bins (1) having a combined capacity of 750 tons. From the bins it goes to a 15 by 30 in. Farrel jaw breaker (3), by way of a shaking-screen feeder (2) with openings 1.5 in. square, and an electro-magnet which removes broken steel, nails, etc. After being crushed to 3-in. maximum size it joins the undersize of screen (2) on its way to elevator (6), which delivers it to a 3 by 8 ft. Symons pulsating screen (7) with $\frac{3}{4}$ -in. round openings. Oversize passes under an electro-magnet, then through a Symons 36-in., style C disk crusher (8), where it is reduced to $\frac{3}{4}$ -in. maximum size. It then joins the undersize of screen (7) on its way to belt conveyor (9).

The conveyor, 630 ft. long, is in two sections, each driven by a separate 20-hp. motor. It travels at a speed of 196 ft. per minute. The vertical lift from bottom to top is 220 ft.

2. Supply Bins and Feeders

On reaching the top of the mill, the ore is dumped into a secondary supply bin (10) of 1,200 tons capacity. From this bin it goes to four 24-in. apron feeders (11), and thence via a horizontal 20-in. belt conveyor (12) to an inclined belt conveyor (13).

Conveyor (13) delivers the ore, ranging from $\frac{3}{4}$ -in. maximum size to the finest material coming from the mine, to two 54 by 18 in. Garfield rolls (14) and (16), which reduce it to 3-mm. maximum size. It then goes by way of No. 1 bucket elevator (15) to four Colorado Iron Works impact screens (17) with openings 2.5 mm. square. Oversize returns to rolls. Undersize goes via an eight-compartment mechanical distributor (20) to any number or all of eight No. 8 Wilfley roughing tables (21).

Principal Machines and Accessories in the Concentrator of the Timber Butte Milling Co., Shown on Flow Sheet, Fig. 3.

Flow Sheet Number	Number of Pieces	Description of Machine
1	2	Crude-ore bins, 750 tons combined capacity.
2	1	Shaking-screen feeder, 1½-in. square openings.
3	1	15 by 30 in. Farrel jaw crusher, crushing to 3 in.
6	1	Steel elevator, 32-ft. centers—95 ft. per minute.
7	1	3 by 8 ft. Symons pulsating screen, 375 strokes per minute, ¾-in. round openings.
8	1	Symons 36-in., style C disk crusher, 133 and 300 rev. per minute, crushing to ¾ in.
9	1	24-in. inclined belt conveyor, 196 ft. per minute.
10	1	Secondary ore bin, 1,200 tons capacity.
11	4	24-in. steel apron feeders, 5 ft. per minute.
12	1	20-in. horizontal belt conveyor, 250 ft. per minute.
13	1	20-in. inclined belt conveyor, 250 ft. per minute.
14	1	54 by 18 in. Garfield rolls, 83 rev. per minute.
15	1	No. 1 belt elevator, 84-ft. centers, 415 ft. per minute.
16	1	54 by 18 in. Garfield rolls, 83 rev. per minute.
17	4	C.I.W. impact screens, 50 strokes per minute, 2½-mm. square openings.
20	1	Eight-compartment mechanical distributor, 21 rev. per minute.
21	8	No. 8 Wilfey roughing tables, 243 strokes per minute.
22	2	45-in. Akins classifiers, 5 rev. per minute.
23	1	No. 2 belt elevator, 72-ft. centers, 400 ft. per minute.
24	1	No. 3 belt elevator 50-ft. centers, 400 ft. per minute.
25	1	Dewatering box.
26	1	Three-compartment distributor.
27	3	8-ft. by 30-in. Hardinge mills, 28 rev. per minute.
28	1	No. 4 belt elevator, 58-ft. centers, 400 ft. per minute.
29	1	Eight-compartment mechanical distributor, 21 rev. per minute.
30	8	No. 3 James sand tables, 262 strokes per minute.
31	1	No. 5 belt elevator, 80-ft. centers, 400 ft. per minute.
32	1	Five-spigot Richards hindered-settling classifier, 40 tons capacity.
33	1	Double, three-compartment Hars jig.
34	2	No. 6 Wilfey tables, 240 strokes per minute.
35	1	No. 3 James sand table, 262 strokes per minute.
36	1	Distributing box.
37	2	45-in. Akins classifiers, 5 rev. per minute.
38	2	8-ft. by 30-in. Hardinge mills, 28 rev. per minute.
39	1	No. 6 belt elevator, 58-ft. centers, 490 ft. per minute.
40	1	16 by 10 ft. sludge tank, 10 rev. per minute.
43	1	Standard Minerals Separation 11-cell flotation machine, capacity 600 tons.
44	1	Standard Minerals Separation 8-cell flotation machine, capacity 200 tons.
45	1	Air lift.
48	1	4½-ft. Hardinge mill, 32 rev. per minute.
49	1	No. 8 belt elevator, 58-ft. centers, 490 ft. per minute.
50	3	45-in. Akins classifiers, 5 rev. per minute.
51	1	No. 7 belt elevator, 81-ft. centers, 490 ft. per minute.
52	1	Esperanza classifier, 25 ft. per minute.
53	1	Five-spigot Richards hindered-settling classifier, 100 tons capacity.
54	7	No. 3 James sand tables, 265 strokes per minute.
55	1	30-in. Akins classifier, 5 rev. per minute.
57	1	36-ft. Dorr thickener, ⅓ rev. per minute.
58	1	Cameron steam pump.
59	1	Cameron steam pump.
60	1	36-ft. Dorr thickener, ⅓ rev. per minute.
61	1	Two-stage centrifugal pump 1½ in.
62	1	10 by 10 ft. hot-water tank.
63	1	45-in. Akins classifier, 5 rev. per minute.
65	2	28-ft. Dorr thickeners, ⅓ rev. per minute.
66	1	No. 9 belt elevator, 52-ft. centers, 440 ft. per minute.
67	1	14 by 10 ft. agitating tank.
68	1	8 by 15½ ft. Montejú tank.
69	1	Kelly filter press, type 850.
70	1	Two-stage centrifugal pump.
71	2	Copper-concentrates bins.
72	2	Lead-concentrates bins.
73	2	Coarse-zinc concentrates bins.
74	2	Fine-zinc concentrates bins.
75	1	30,000-gal. dirty-water tank.
76	1	30,000-gal. fresh-water supply tank.
77	1	50,000-gal. sprinkler tank for fire protection.
78	2	Settling boxes in concentrate house.

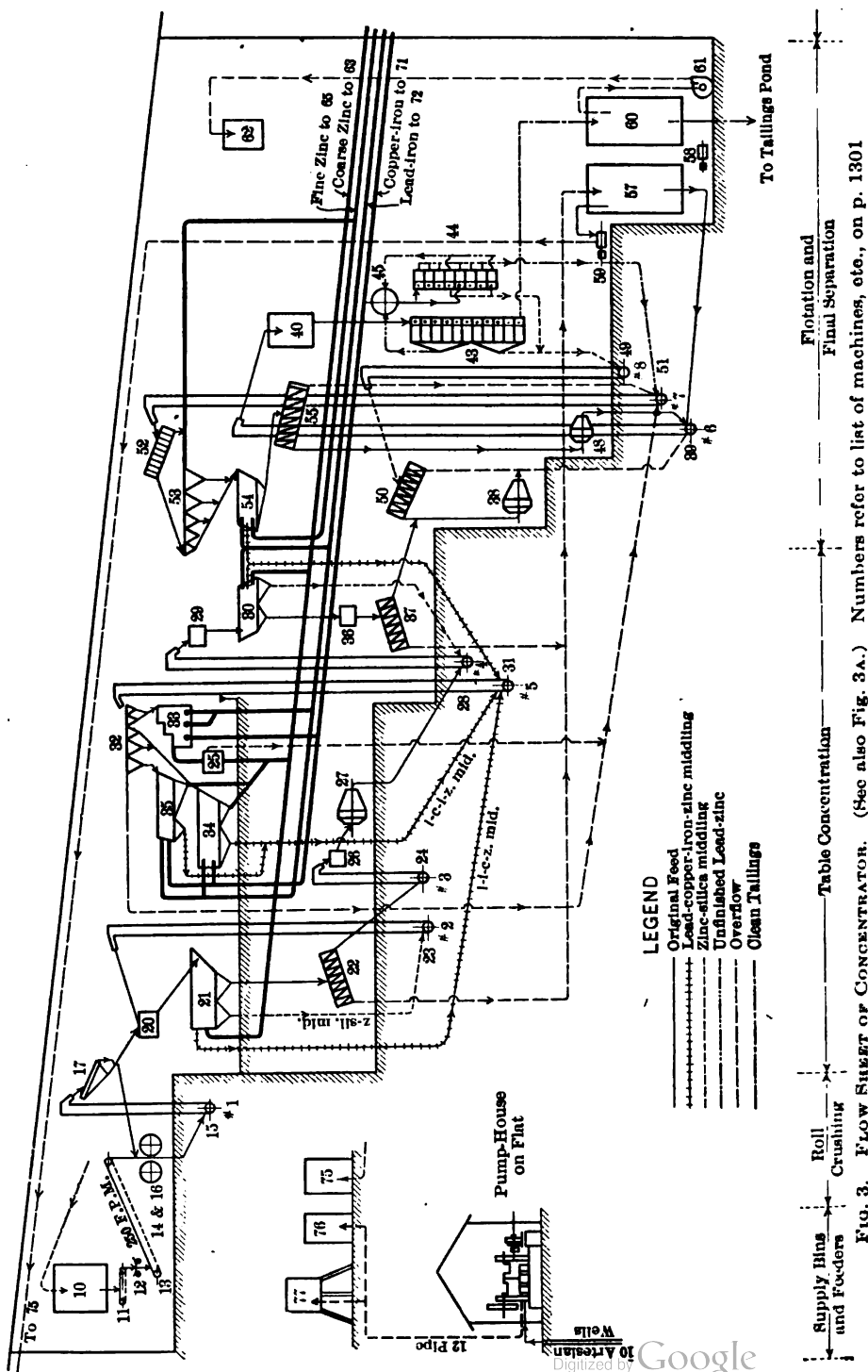


FIG. 3. FLOW SHEET OF CONCENTRATOR. (See also Fig. 3A.) Numbers refer to list of machines, etc., on p. 1301

3. *Table Concentration*

Only four of these tables are used at present, each treating on an average 100 tons of crude ore per day, and also handling its proportion of circulating middlings. Each table makes the following products:

1. A lead-copper-iron-zinc middling, which goes to No. 5 elevator (31) for further treatment (see "Description of Processes").

2. A coarse-zinc concentrate, which goes to bin (73) via classifier (63). In order to keep this product at all times as free as possible from silica it is cut off at a safe point near the lower end of the concentrates discharge. This allows some of the free zinc to go over into the next product.

3. A zinc-silica middling—so called—which is returned to the same tables via No. 2 elevator (23) and distributor (20). In this manner the free zinc that was permitted to mix with this product eventually finds its way into the zone in which it belongs, while the true middling particles join the tailings product No. 4 to go to Hardinge mills (27) for regrinding.

4. Tailings and slimes, which pass to two 45-in. Akins classifiers (22) for further treatment.

Product No. 1 is delivered by No. 5 elevator (31) to a five-spigot Richards hindered-settling classifier (32), the overflow from which goes to No. 7 elevator (51). The first two spigots of the classifier discharge into a double three-compartment Harz jig (33). Only one side is used at present. The jig makes the following products:

1. A finished lead concentrate from cup and hutch of first two compartments, which goes to bin (72).

2. A finished copper-iron concentrate from cup and hutch of third compartment, which goes to bin (71).

3. The tail overflow is a coarse-zinc concentrate which, after being partly dewatered in box (25), goes to bin (73) via classifier (63). The overflow of dewatering box (25) goes to No. 7 elevator (51).

The third and fourth spigot product of classifier (32) goes to two No. 6 Wilfley tables (34); the fifth spigot feeds one No. 3 James sand table (35). Wilfleys and James table each make four products:

1. A lead-iron concentrate, which goes to bin (72).

2. A copper-iron concentrate, which goes to bin (71).

3. A copper-iron-zinc middling—so called—which is returned to same tables via No. 5 elevator (31) and classifier (32).

4. Tailings, which is a coarse-zinc concentrate, and goes to bin (73) via classifier (63).

Additional tables are being installed for this work, which will result in a better separation of the minerals.

Tailings from the Wilfley roughing tables (21) go to two 45-in. Akins classifiers (22), which overflow to a 36-ft. Dorr thickener (57). Sands go

4. A zinc-silica middling, which contains some of the free zinc of the preceding product, and is therefore returned to the same tables via No. 4 elevator (28) for the reasons explained for the Wilfley roughers (21).

5. Tailings, which go via distributing box (36) to two 45-in. Akins classifiers (37). Overflow goes to Dorr thickener (57). Sands to two 8-ft. by 30-in. Hardinge mills (38), where it is attempted to crush to a maximum size of 60 mesh. Product is delivered by No. 6 elevator (39) to a 16 by 10 ft. sludge tank (40) in the flotation section.

4. Flotation Section and Final Separation

In the flotation section, the final separation of the fine sulphide minerals from the gangue takes place.

For a satisfactory separation acid (H_2SO_4) and heat are found necessary adjuncts to the generally accepted oil-flotation principles.

From the sludge tank (40), which acts as an equalizer between the gravity-concentration and flotation section, the pulp flows to a standard 600-ton Minerals Separation unit (43) known as the "rougher machine." It consists of eleven cells, each divided into a mixing or agitating and a flotation compartment. The intensely agitated pulp discharges from the mixing into the flotation compartment, where the pulp is comparatively still and where a portion of the sulphide minerals rises to the surface to be skimmed off by revolving paddles and discharged into a launder in front of the cells. The remainder of the sulphides and the gangue that has settled to the bottom of each flotation compartment are drawn into the adjoining cell by the pumping action of its agitator. Here the process is repeated and sulphides not floated in the first cell are given a chance in the second cell, and so on.

The overflow from the last five cells of the "rougher machine" (43) is a middling that goes via No. 8 elevator (49) to Akins classifiers (50). The classifier overflow returns to the "rougher machine" of the flotation section. Sands are reground in two 8-ft. by 30-in. Hardinge mills (38) to an approximate 60-mesh maximum size and return via No. 6 elevator (39) to flotation section.

Finished tailings from the eleventh cell of the "rougher machine" (43) flow to a 36-ft. Dorr thickener (60) to be partly dewatered and thence to the tailings pond. The overflow from this tank is hot water, free from solids. It is pumped by a two-stage centrifugal pump (61) to a 10 by 10 ft. tank (62), which supplies the flotation machines with hot water.

The mixed sulphide concentrate which overflows from the first six cells of the "rougher machine" (43) goes via air lift (45) to a 200-ton Minerals Separation unit (44), consisting of eight cells, known as the "finisher." Here the "rougher" concentrates are refloatated for the pur-

pose of removing more of the gangue minerals. Cells Nos. 1-2-3-5-6-7 of this machine (44) make a float sulphide which goes via No. 7 elevator (51) to classifiers and James tables for final separation of lead and iron from the zinc.

Overflow from cells 4 and 8 of "finisher" (44) returns via air lift (45) to the same unit. The bottom discharge from cells 4 and 8 is a middling which joins the middlings from the "rougher machine" (43) to be reground in Hardinge mills (38) and returned to the flotation section.

The float concentrates from unit (44) are lifted by No. 7 elevator (51) to an Esperanza classifier (52), the overflow from which is a finished fine-zinc concentrate that goes to two 28-ft. Dorr thickeners (65) in the concentrate building. Sands go to a five-spigot Richards hindered-settling classifier (53), whose overflow is a finished fine-zinc concentrate, which joins the overflow of the Esperanza classifier.

The discharge from the five spigots of the Richards classifier is distributed on seven No. 3 James sand tables (54), where the lead and zinc are separated and the insolubles in the zinc concentrate reduced to about 3.5 per cent. Each table makes the following products:

1. A finished lead concentrate, which goes to the bin (72).
2. A lead-copper-iron-zinc middling, which is returned via No. 5 elevator (31) to the primary jig and table re-treatment circuit.
3. A fine-zinc concentrate, which goes to Dorr thickeners (65) in the concentrate building to be partly dewatered.
4. A tailing, which is reground in a 4½-ft. Hardinge mill (48) after passing through a 30-in. Akins classifier (55) whose overflow returns via No. 7 elevator (51) to the final table-separation circuit.

The product of the Hardinge mill (48), which is of about 60-mesh maximum size, goes to flotation section via No. 6 elevator (39).

The Dorr thickener (57) receives practically all the fines and slimes coming from the mine and those made in the crushing operations. The overflow is dirty water, which is pumped by a double Cameron steam pump (59) to the "dirty-water tank" (75) at the top of the mill, whence it returns for circulation throughout the mill. A duplicate auxiliary Cameron pump (58) is put into service whenever pump (59) is stopped for repairs. From the bottom of tank (57) the thickened pulp is lifted by hydrostatic pressure to the boot of No. 6 elevator (39) to go to flotation section.

5. Concentrate House

The chief operation carried on in this department is the dewatering of the fine-zinc concentrates made in the flotation and final separation section. These concentrates are all collected in two 28-ft. Dorr thickeners (65), the overflow from which is practically clear water. This goes at present to waste, joining the tailings from tank (60) in the flota-

tion section. The thickened concentrates, carrying about 30 per cent. moisture, are discharged at the bottom of the Dorr thickeners (65) and go via No. 9 elevator (66) to a 14 by 10 ft. agitating tank (67), and from there to an 8 by 15½ ft. Montegu tank (68) that carries 80 pounds air pressure. From this tank they are forced into a Kelly filter press (69), 850 type, where the moisture is reduced from 30 to about 10 per cent. The liquid from the press is practically clear water and returns to the Dorr thickeners (65), while the cake drops into the fine-zinc concentrate bins (74) whence it is drawn into the railroad cars for shipment to the smelter.

The coarse-zinc concentrates upon entering this building pass through a 45-in. Akins classifier (63) to be dewatered, before going to the coarse-zinc concentrate bins (73). The overflow from classifier (63) returns to the Dorr thickeners (65).

The copper-concentrate bins (71) and the lead-concentrate bins (72) are each provided with a filter bottom through which much of the moisture in the concentrates is drained and passed to two settling tanks (78), the overflow from which is pumped back to the Dorr thickeners (65) by a two-stage centrifugal pump (70).

The coarse (73) and fine zinc (74) concentrate bins also have filter bottoms and drain, whenever there is sufficient moisture in bin products, direct to the sump of centrifugal pump (70).

III. OPERATING DETAILS

1. *Capacity, Efficiency, and Economy of Mill*

Since operations began in June, 1914, the quantity of crude ore treated has been increased from 350 to 450 tons per day. The ratio of concentration of crude ore into zinc concentrates in round figures is three into one. More exactly, for the month of September, 1914, it was 2.89 into one. At this ratio 450 tons of ore treated per day would result in 150 tons of zinc concentrates.

Roughly, 74 per cent. of this amount, or 111 tons, is fine zinc, including concentrates from flotation, final separation, and the fine portions of the table concentrates.

The ratio of concentration of crude ore into lead and copper-iron products is approximately 125 into one, 20 per cent. of which is lead concentrates.

Treatment costs have averaged between \$2 and \$2.25 per ton of ore passed through the mill. This covers labor, power, water, supplies, repairs, and general expenses, but not interest on investment and depreciation.

TABLE I.—*Average Analysis of Crude Ore, Concentrates, and Tailings*

Name of Product	Cu., Per Cent.	Ag., Oz. per Ton	Au., Oz. per Ton	Fe., Per Cent.	Zn., Per Cent.	Pb., Per Cent.	Insol., Per Cent.
Original ore.....	0.73	6.23	0.013	3.29	18.52	0.75	62.23
Copper-iron concentrates	2.25	21.01	0.04	19.78	22.78	13.01	2.30
Lead-iron concentrates..	1.13	18.91	0.04	9.99	9.15	50.60	5.05
Coarse-zinc concentrates	1.96	14.74	0.035	9.15	47.77	1.88	3.47
Fine-zinc concentrates..	2.22	16.84	0.036	4.45	52.44	1.72	5.05
Tailings.....	0.07	0.60	0.0017	0.44	1.20	0.05	95.42

A comparison of the tonnage of concentrates and their analysis with the tonnage and analysis of the crude ore milled, shows an extraction of about 96 per cent. of the zinc and about 92.6 per cent. of the silver contents of the original ore.

The copper-iron concentrates go to the Washoe works at Anaconda, Mont.; the lead concentrates to the lead smelter of the United States Smelting Co. at Midvale, Utah; and the zinc concentrates to the zinc smelters at Bartlesville, Okla., and Argentine, Kan.

2. Description of Processes

A. Gravity-Concentration Section.—The most striking feature in this section of the mill is the performance of the Wilfley roughing tables. They are the first concentrating machines encountered in the mill and take the undersize from the impact screens without any previous classification. This material ranges from 2.5 mm. maximum diameter to the finest stuff in the ore. The tables not only handle an unusual quantity of material, amounting to more than 100 tons per table daily, but they make at the very outset a finished coarse-zinc concentrate, containing from 49 to 51 per cent. zinc and not more than 4.0 per cent. insolubles. Moreover, they also send all the fines coming from the mine and those made in the preceding crushing operations to two Akins classifiers to be separated from the sands, after which they go directly to the M. S. flotation section where they belong. Thus none of the intermediate machines are incumbered with this troublesome material, and the losses incident to frequent handling of slimes are reduced to a minimum. Moreover, the tables furnish as their first product a so-called lead-copper-iron-zinc middling which by the elimination of most of the silica has been enriched to a point where it can be profitably re-treated.

This product No. 1 is taken from the portion of the tables where the heavier-than-blende sulphides collect. With the Elm Orlu ore this results in a concentration of the lead-iron and copper sulphides at this point. The "cut" of product No. 1 is made at a point which will not only prevent excessive amounts of the mixed lead-copper-iron sulphides

from entering and contaminating the next following product No. 2, intended as a high-grade finished zinc concentrate, but will also prevent the remaining of excessive amounts of free zinc blende in product No. 1.

As regards gangue, product No. 1 is virtually a concentrate in which the insoluble has been reduced to the economic limit. It might be called a "combination of concentrates," being in reality a mixture of sulphides of the base metals. As such it has no commercial value; and, pending further treatment, it is considered a "middling" in the sense of being an unfinished product.

Considering the zinc blende in this product as "the" sulphide and the other sulphides as the gangue, all conditions for a true middling would prevail; i.e., a mechanical mixture of the finer, already free sulphides up to the coarser sizes, consisting of the metallic sulphides interlocked with gangue minerals. In the case under consideration, the interlocked minerals are in amounts too small to warrant regrinding, an operation usually associated with the conception of a middling product.

From the original mill feed, containing only a small percentage of lead and copper, it is practically impossible to make commercial products of those metals by gravity methods in one operation, without a large installation of machinery and without high treatment charges. However, by the roughing operations it is possible to make this rough middling product No. 1, which, due to the enrichment and to the relatively small tonnage of the product, becomes amenable to final economical separation by gravity processes. Thus the taking off of the No. 1 product from the roughing tables results: first, in a much higher grade of finished zinc concentrate by the exclusion of other sulphides; and second, in making possible the economic gravity separation of these sulphides into a finished lead concentrate and a finished copper-iron concentrate.

All this is accomplished by a clever modification and distribution of the riffles, by giving the tables a rather steep inclination (8° to 10°), and by adding the proper quantity of wash water.

The remarks made here about this product No. 1 apply with equal force to the lead-copper-iron-zinc middling product made on the James roughing tables (30).

B. "*Minerals Separation*" *Flotation Section*.—Flotation processes like the one employed in this mill present such unusual possibilities in their adaptation to the treatment of metalliferous ores that the study of a process which has successfully solved the problem of treating a particular ore is of the highest interest. Although, in America, flotation has so far been most successfully employed in the treatment of sulphide ores, it seems highly probable that when the interrelation of the underlying chemical and physical laws is more fully understood flotation will be extended with equal success to the treatment of oxide and carbonate ores.

The simplicity of the processes and of the machinery employed; the small floor space occupied by such machinery compared with old-type concentrators of the same capacity; the fact that they accomplish a separation where other processes have failed; and their many other advantages, are so striking that it looks to be merely a question of time when flotation will not only usurp many of the functions of the older gravity processes but will eventually supplant them altogether, at least for the treatment of the finer material and of slimes.

In view of the scientific and commercial importance of flotation, a brief outline of the plant installed at the Timber Butte mill is given in the following pages:

To unlock the finest included grains of sulphides from the adhering gangue, prior to separation, it is necessary to crush the Elm Orlu ore to a size between 40 and 60 mesh maximum. In the course of tests and experiments, preceding the design and building of the mill, it was found that attempts at extracting the zinc from material finer than 40 mesh by a simple process of gravity concentration, resulted in a low grade of concentrates, which could only be increased by a costly installation.

This led to the adoption of "flotation" according to the process of the Minerals Separation Co. In this process use is made of the now well-known facts that when an ore containing metallic sulphides is mixed with water, oil, and acid and agitated together in proper proportions, the oil not only has a selective action for the sulphides, but the air bubbles formed by agitation have a selective action for the oiled sulphide particles, to which they attach themselves and thus float them to the surface in the shape of a froth which overflows or is mechanically skimmed off. The gangue minerals, on the other hand, have a comparatively feeble adhesiveness to gas films and oil, and a strong adhesiveness to water, which is greatly increased by a slight acidulation of the water. For this reason they quickly become wetted and sink to the bottom of the flotation machine, resulting in the desired separation of the sulphides from the gangue.

No satisfactory theory has as yet been propounded, as to why acid promotes the preferential adhesion of water to gangue and also probably increases at the same time the preferential adhesion of oil to sulphides. For commercial purposes it suffices to recognize the facts and make practical use of them.

This is done in the Minerals Separation process by passing the mixed pulp through one or more standard machines, consisting of a series of cells which have the shape outlined in Fig. 4. The pulp is kept at a uniform density in the proportion of three of water to one of ore, and at the Timber Butte mill is heated to a temperature of 130° F. by the introduction of live steam. From the tests it appeared that with the Elm Orlu

ore higher temperatures did not produce better results, while temperatures below 125° F. reduced the efficiency of the plant.

One of the effects of heating the pulp is the minimizing of the force of adhesion which tends to surround the mineral particles with water; that is, to wet them and thereby detach the air bubbles that are instrumental in floating them to the surface.

Sulphuric acid is mixed with the pulp at the rate of 7 to 8 lb. of acid to every ton of original ore going through the mill. Oil is added in the proportion of 0.5 lb. per ton. In the Timber Butte mill, the pulp is first passed through a standard 600-ton Minerals Separation unit (43) consisting of eleven cells. This is known as the "rougher machine," because it is used, not to make a finished product, but rather to treat a maximum amount of material "roughly;" that is, to get rid of all clean tailings and save for re-treatment all the sulphides and the more or less

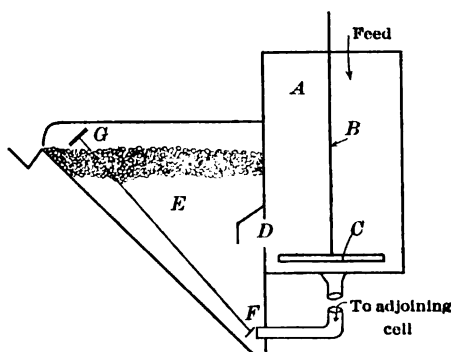


FIG. 4.—OUTLINE OF M. S. FLOTATION CELL.

rich middlings. At the Timber Butte mill, oil is introduced in the mixing compartment (A) of the first cell where the pulp is violently churned by the action of the agitator (B), which at this plant makes 265 rev. per minute and causes the propellers (C) to run at a peripheral speed of about 1,500 ft. per minute. The aerated mixture passes through the opening (D) into the flotation compartment (E), where the froth containing the sulphides of lead, copper, zinc, and iron, as well as some of the richer middlings, immediately rises to the surface, while the clean gangue and the leaner middlings drop to the bottom.

The flotation compartment has the shape of a spitzkasten, the apex of which is provided with a valve (F), adjustable by a hand wheel (G). Through this valve the material that has settled to the bottom of the first cell is drawn into the adjoining cell by the pumping action of its propeller. The process is here repeated and sulphides which did not float in the first cell are given a chance to do so in the second one, and so on. By the time the pulp has reached the seventh cell of the "rougher

machine" the clean sulphides and the richer middlings have all been floated. The remaining middlings and tailings in their passage from cell No. 7 to cell No. 11 are separated from each other by the successive floating of the middlings until in the bottom of the last cell (No. 11) only clean tailings remain, which are taken to a Dorr thickener (60) for extraction of the hot water. This is pumped to a tank (62) to be re-used in the flotation machines.

The middlings overflowing from cells 7 to 11 are reground in Hardinge mills (38) and returned to the flotation section.

The float concentrate from the first six cells of the "rougher machine" (43) is a product rich in zinc (about 47.8 per cent.), but due to the presence of middlings still contains about 14 per cent. insolubles. It is therefore sent to the first and fifth cells of a 200-ton Minerals Separation flotation unit (44) containing eight cells. This is known as the "finisher machine" because it makes a finished sulphide concentrate. On passing the pulp through the successive cells of the "finisher," the sulphides are floated to the surface while the clean middlings ultimately reach the bottom of the fourth and eighth cells; whence they go to Hardinge mills (38) to be reground and returned to the flotation section.

The float concentrate from cells 1-2-3-5-6-7 of the "finisher machine" (44) is a finished sulphide product containing, besides the sulphides of lead, copper, and iron, 52 per cent. zinc and 3.5 per cent. insolubles. The fines in this product overflow as fine-zinc concentrate from an Esperanza and a Richards classifier through which the product is passed, the coarser portion being re-treated on James tables (54). On these tables the lead and iron are separated from the zinc and the insolubles in the zinc concentrate are reduced to 3.5 per cent.

The overflow from cells 4 and 8 of the "finisher machine," being apt to contain a varying amount of middlings, is returned via air lift (45) to the same machine.

The agitating compartments of the flotation cells are lined with cast iron, and the propellers are made of brass. All agitators of a standard machine are driven from one main shaft by gears which are protected from dust by steel housings. Gears are placed so that propellers in adjoining cells revolve in opposite directions, thus equalizing the strains on the machine and the power transmission.

IV. POWER

1. *Electric Power*

Electric power is furnished over a 2-mile transmission line from the substation of the Montana Power Co. on Montana Street near the old Butte reduction works. Three Westinghouse 350 K.V.A. transformers,

housed in a fireproof concrete building, reduce the voltage from 3,800 to 460 volts for use in the mill motors.

Individual motors are used to drive individual machines or sets of machines.

List of Electric Motors

Motor No.	Horse-power	Machinery Driven by Motors	Machine's Number on Flow Sheet
1	150	2 sets of Garfield rolls.....	14 and 16
		4 apron feeders.....	11
		2 belt conveyors.....	12 and 13
		No. 1 elevator.....	15
		4 impact screens.....	17
2	50	8 Wilfley roughing tables.....	21
		No. 4 elevator.....	28
		No. 5 elevator.....	31
		Double three-compartment jig.....	33
		2 Wilfleys.....	34
		1 James table.....	35
		1 distributor.....	20
3	20	No. 2 elevator.....	23
4	50	1 Hardinge mill.....	27
5	50	1 Hardinge mill.....	27
6	50	1 Hardinge mill.....	27
7	7½	2 Akins 45-in. classifiers.....	22
8	7½	2 Akins 45-in. classifiers.....	37
9	20	No. 3 elevator.....	24
10	40	8 James roughing tables.....	30
		7 James finishing tables.....	54
		No. 7 elevator.....	51
		Esperanza classifier.....	52
		Mechanical distributor.....	29
11	50	1 Hardinge mill.....	38
12	50	1 Hardinge mill.....	38
13	30	No. 6 elevator.....	39
14	30	No. 8 elevator.....	49
15	20	1 Hardinge mill.....	48
		3 Akins 45-in. classifiers.....	50
		1 Akins classifier.....	55
16	75	Rougher flotation cells.....	43
17	25	Finisher flotation cells.....	44
18	7½	2 Dorr thickeners and sludge tank.....	57, 60, and 40
19	25	1 double centrifugal pump.....	61

List of Electric Motors (Continued)

Motor No.	Horse-power	Machinery Driven by Motors	Machine's Number on Flow Sheet
20	125	1 shaking screen.....	2
		1 Farrel crusher.....	3
		1 Symons screen.....	7
		1 Symons crusher.....	8
		1 steel elevator.....	6
		1 Ingersoll two-stage compressor.....	
21	25	2 Dorr thickeners.....	65
		1 Akins classifier.....	63
		No. 9 elevator.....	66
22	20	1 centrifugal pump.....	70
23	20	Belt conveyor, lower section.....	9
24	20	Belt conveyor, upper section.....	9
25	50	Clayton two-stage compressor.....	
26	37	Tramway hoist.....	
27	20	Vacuum pump in pumping station.....	
28	150	Triplex pump.....	

A number of smaller motors are employed in and about the mill for grinding samples, for running tools in the shops, etc.

From meter readings, the consumption of electric power during the month of October, 1914, was as follows: In the mill, 803.6 hp., or 1.99 hp. per ton of ore milled. At the pumping station, 120.2 hp.

2. Steam Power

Steam required in the flotation section for heating the pulp and for running the Cameron pumps is furnished by two 200-hp. Springfield boilers, while one 150-hp. boiler supplies steam for the radiators in the mill and other buildings. The boilers are all connected with each other and the steam is carried to the various buildings in concrete underground conduits.

3. Compressed-Air Power

An Ingersoll-Rand, Imperial type, two-stage compressor with a capacity of 240 cu. ft. of free air per minute, and a Clayton two-stage compressor with a capacity of 300 cu. ft. per minute, supply air for operating the air gates at the ore bins and the Kelly filter press; for pumping the acid from the railroad tanks into the mill tank; for various tools in the repair shops, and for the air lift in the flotation section.

V. WATER SUPPLY

Fresh water for milling purposes is pumped to a 30,000-gal. tank (76) at the top of the mill from the company's pumping station on the Flat, about 2 miles northeast of the mill. At this plant 10 artesian wells, ranging between 38 and 59 ft. in depth, are connected on top to a duplex 12 by 12 in. vacuum pump, driven by a 20-hp. electric motor. The vacuum pump delivers the water into a sump underneath a Platt Iron Works triplex 11 by 15 in. pump of 750 gal. capacity per minute, driven by a 150-hp. electric motor. This pump raises the water in one lift through a 12-in. pipe to the mill tank (76), 450 ft. above the pump station, whence it flows by gravity to the various sections of the mill. Fresh water is used for wash water on Wilfley and James tables, also in jig and in two Richards hydraulic classifiers.

Clear water for future requirements is also available from the overflow of the two Dorr thickeners (65) in the concentrate house. This overflow, which amounts to about 530 gal. per minute, at present goes to the tailings pond. In case of failure of the pumping plant on the Flat to do its work, the fire pump at the mill can be started up to supply the mill tank with city water, to which the pump is connected.

Dirty water, amounting to about 130 gal. per minute, overflows from the Dorr thickener (57) in the flotation section. It contains 0.9 per cent. solids and is pumped by a Cameron steam pump to the dirty-water tank (75) at the top of the mill, whence it returns by gravity to the various sections of the mill. This water is used in rolls, on impact screens, on the feed end of Wilfley and James tables, in Hardinge mills, etc.

Hot water overflows from the Dorr thickener (60) at a variable rate. It is used in the flotation machines (43 and 44), being supplied from a hot-water tank (62), to which it is pumped by a centrifugal pump (61.)

VI. FIRE PROTECTION

A complete system of fire protection has been installed throughout the mill and surrounding buildings. It consists of the automatic sprinkler system of the International Company of Philadelphia, and is supplied with water from a 50,000-gal. tank (77) at the top of the mill. This tank is kept filled and used only for fire purposes. A set of steam coils keeps the water from freezing in cold weather. A by-pass in the main pipe line supplying fresh water to the mill tank (76) from the pumping station on the Flat, also supplies the sprinkler tank (77), which is located at an elevation high enough to send a stream of water by hydrostatic pressure to the highest point of the mill. Thirteen hydrants are scattered over the premises, in convenient location to cover all portions of the plant (see Fig. 2).

If for any reason the water supply from the pumping station drops below a certain limit and pressure, a large fire pump is automatically started which supplies the fire circuit with water from the system of the Butte Water Works.

The fire pump is a triplex 11 by 15 in. Platt Iron Works pump, driven by a 125-hp. electric motor and having a capacity of 750 gal. per minute against 200 lb. pressure. The pump is housed in a concrete, fireproof building, at an elevation which permits the sump underneath it to fill with city water by hydrostatic pressure as fast as it is taken out.

VII. CONCLUSIONS

A visitor to the Timber Butte mill cannot help being impressed at once by the up-to-date appearance of the plant and the completeness with which every detail has been worked out that makes for efficiency, economy, and comfort. There is room in abundance for all machinery. Every part of it is accessible and illuminated in daytime by a wealth of natural light and at night, by a liberal distribution of large tungsten lamps.

Shafts and belts are arranged so as to minimize the danger to attendants and at the same time be easily accessible for lubricating and necessary repairs. Launderers, elevators, conveyors, are systematically arranged and open for inspection throughout their whole length.

Numerous steel radiators keep the mill comfortably warm in cold weather and a large fan in the flotation section frees the atmosphere from excessive moisture due to condensation of steam.

A spacious modern office building and a large assay office supplied with all modern conveniences add materially to the completeness and appearance of the whole plant, every detail of which manifests the foresight of the company and the skill of its engineers.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Cyaniding Practice of Churchill Milling Co., Wonder, Nev.

BY E. E. CARPENTER,* E. M., WONDER, NEV.

(San Francisco Meeting, September, 1915)

BELIEVING that the results accomplished in the mill of the Churchill Milling Co., Wonder, Nev., during the fiscal year ended Sept. 30, 1914, will be of interest, I am presenting the more prominent facts and figures showing the progress made in treating this ore with especial reference to tonnage, supplies, costs, and extractions.

The mill is situated at Wonder, Nev., in the Wonder mining district, 55 miles east from Fallon, Nev., the nearest railroad point. All supplies are hauled this distance by freight teams, this method of transportation having been found more reliable for continuous service than trucks, the price paid for hauling being slightly under one cent per pound. In this mill are treated the ores of the Nevada Wonder Mining Co.

Ore is Unusually Hard

The company which tested this ore before the mill was built reported "This ore is an unusually hard, tough, oxidized quartz, carrying a small percentage of high-grade sulphides, containing good values in both gold and silver. None of the rebellious silver minerals appear to occur in appreciable quantities." . . . "In regard to the mill tonnage, this ore is unusually hard and tough, and a high stamp duty cannot be expected. Using a 20-mesh screen and 1,050-lb. stamps, our best work was three tons per stamp per 24 hours."

It is very probable that the ore delivered to the testing plant was harder than the regular mill run has proved to be. However, at present the mine is developed to a depth of 1,000 ft., and still shows hard ore in places at this depth in about the same proportion as on the upper levels from which the test ore came; during this year's work, ore was treated from nearly all levels.

Together with this hard ore in the mill feed a considerable proportion appears to be broken down and tending to crumble, thus making an ideal mixture for stamp milling. A small proportion of talc is also noticeable, which sometimes increases in amount to the point where it is very troublesome in the thickeners.

* Superintendent, Churchill Milling Co.

Mill Capacity 133 Tons

The mill was designed and built as a 75-ton mill and cyanide plant on a 30° slope, the equipment being as follows, named in order of use:

- 1 10 by 16 in. Blake crusher.
- 10 1,400-lb. stamps.
- 1 6 ft. Trent Chilean mill.
- 1 Dorr duplex classifier,
- 1 5 by 22 ft. tube mill.
- 1 24 by 14 ft. Dorr thickener.
- 4 15 by 45 ft. Pachuca-type agitators.
- 1 28 by 10 ft. Dorr thickener.
- 1 28 ft. by 14 ft. stock tank with stirrer.
- 2 8 by 11 ft. 6 in. Oliver filters.
- 5 Zinc boxes.
- 2 Melting furnaces.

The necessary equipment of solution pumps, slime pumps, vacuum pump, boiler, compressor, gold tanks, battery storage tank, sump tank, etc., as shown in accompanying flow sheet.

The principal changes made since the mill was built are the addition of one 28 by 10 ft. Dorr thickener, between the agitators and stock tank, required to secure sufficient replacing of solution; one 14 by 11 ft. 6 in. Oliver filter made necessary by the greater tonnage and the talc; and two additional zinc boxes. Later a larger compressor was installed, the old one being held in reserve.

With the exception of these changes and the discontinuation of concentration, which was practiced only for a short time, the mill is being operated practically as built. The tonnage has been gradually increased until this year it has averaged 133.1 tons per day or 13.31 tons per stamp per 24 hr. The stamps dropped 92.4 per cent. of the time. On the basis of actual operating time, the rate of crushing was 14.40 tons per stamp per 24 hr. for the year. In September, the tonnage of 4,335 tons for 30 days averages 144.5 tons per day, and, based on actual operating time, 15.45 tons per stamp per 24 hr.

The annual report published in November, 1914, shows that the ore contained 10,515.895 oz. of gold of which 10,005.055 oz. were shipped as bullion; 946,050.49 oz. of silver of which 874,085.49 oz. were shipped as bullion (except for one refinery-slag shipment); showing a recovery of 95.1 per cent. of the gold contents, 92.4 per cent. of the silver contents, and 93.2 per cent. of the gross value. The above figures give the average contents of the mill heads as 0.216 oz. of gold and 19.48 oz. of silver.

The tonnage was arrived at by weighing each car, containing approximately 4 tons, as it came to the mill, and deducting the daily moisture determination, which averaged about 3 per cent. The precious-metal content of the ore, as shown above, was determined by combining the

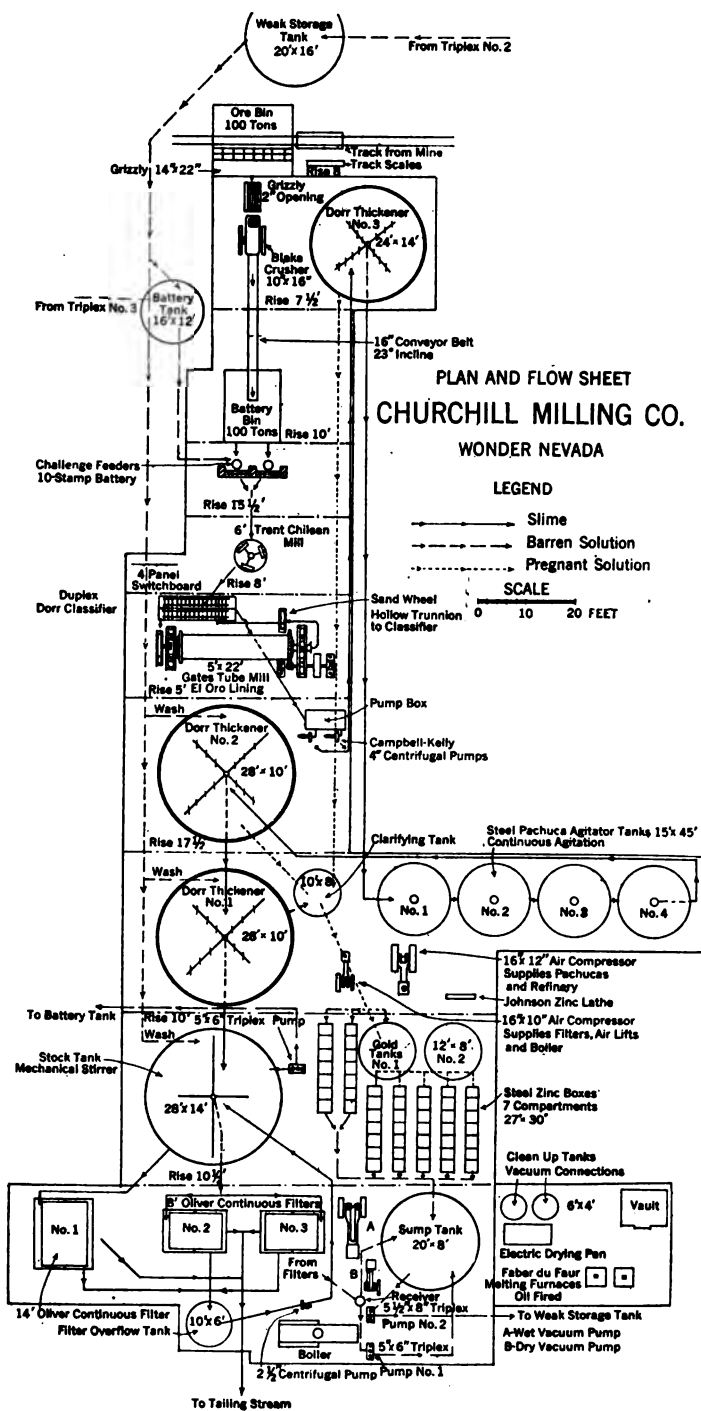


FIG. 1.—FLOW SHEET OF CYANIDE PLANT, CHURCHILL MILLING CO.

bullion shipped and the loss in tailings, the former being smelter returns and the latter the product of the daily tonnage multiplied by the average value of three separate unwashed assays of the tailings for each day, one taken from each filter. Each sample represents the cake discharged from a filter and is the result of a sample taken each half hour from each filter as the discharged cake falls into the sluicing launder; this cut is taken in a dipper which insures catching a proportionate amount of water.

TABLE I.—*Tonnage, Cost, and Recovery Data*

Months	Tons per Month	Tons per Day	Total Direct Cost	Cost to Date	Gold Recovery, Per Cent.	Silver Recovery, Per Cent.	Value Recovery, Per Cent.
Oct., 1913.....	4,160	134.2	\$2.825	\$2.825	95.3	92.8	93.6
Nov., 1913.....	4,032	134.4	2.852	2.838	94.2	92.9	93.7
Dec., 1913.....	4,103	132.4	2.812	2.830	94.3	93.4	93.7
Jan., 1914.....	3,357	108.3	3.189	2.906	95.2	92.1	92.5
Feb., 1914.....	3,254	116.2	2.861	2.898	93.8	93.6	93.7
March, 1914....	4,350	140.3	2.567	2.836	94.8	93.7	93.9
Apr., 1914.....	3,730	124.3	2.604	2.804	95.3	92.2	93.1
May, 1914.....	4,217	136.0	2.598	2.777	94.3	91.1	91.2
June, 1914.....	4,200	140.0	2.521	2.746	95.6	89.4	91.7
July, 1914.....	4,392	141.7	2.781	2.750	96.3	90.6	92.1
Aug., 1914.....	4,440	143.2	2.657	2.739	97.3	93.4	94.3
Sept., 1914....	4,335	144.5	2.771	2.743	97.3	92.7	94.0
Averages or Totals.....	48,570	133.1	2.743	2.743	95.1	92.4	93.2

Average contents of the ore: Gold, 0.216 oz.; silver, 19.48 oz.; valued at \$15.52 per ton.

Table I shows the tons treated each month, the tons per day, the total direct cost, the cost to date for each month, the gold, silver, and value recovery, together with the averages. The direct costs include all charges for labor both in and about the mill, superintendent's salary, mill assaying, storehouse expense, storing tailings, etc.; all charges for supplies including water, chemicals, repairs, replacements, and small items for permanent improvements, and the power cost as measured coming into the transformers where the current is reduced from 55,000 to 440 volts.

This table shows some interesting results in that for only one month of the year did the gold recovery fall below 94 per cent., for two months it reached 97.3 per cent. and this on an ore containing 88.9 oz. of silver for each ounce of gold. The gold recovery varied from 93.8 to 97.3 per cent.; the silver recovery from 89.4 to 93.7 per cent.; the value recovery was from 91.2 to 94.3 per cent.

TABLE II.—Average Costs for Year per Ton of Ore Treated

	Labor	Supplies	Power	Total
Superintendence.....	\$0.074			\$0.074
Crushing and conveying.....	0.045	\$0.024	\$0.012	0.081
Stamps.....	0.081	0.031	0.073	0.185
Chilean.....	0.028	0.053	0.063	0.142
Tube mill.....	0.029	0.119	0.107	0.255
Elevating and separating.....	0.012	0.016	0.010	0.038
Agitation ^a	0.148	0.589	0.133	0.870
Filtering.....	0.079	0.040	0.026	0.145
Precipitation.....	0.089	0.224		0.313
Refining.....	0.049	0.125	0.002	0.176
Assaying.....	0.043	0.022	0.001	0.066
Surface and plant ^b	0.051	0.014	0.014	0.079
General.....	0.005	0.005		0.010
Storehouse.....	0.046	0.002		0.048
Water.....	0.045	0.026		0.071
Steam heating ^c	0.035	0.155		0.190
Total.....	\$0.857	\$1.445	\$0.441	\$2.743

^a Chemicals charged in supplies for agitation were

Cyanide.....	\$0.375
Lime.....	0.110
Lead.....	0.089
	<u>\$0.574</u>

^b Includes expense of storing tailings.

^c Cost of crude oil was 10.7c. per gallon at the mill.

Table II shows the direct costs divided into the various departments, and each department divided into cost for labor, supplies, and power; also the yearly average cost for each department, and the yearly average cost per ton for labor, supplies, and power. The cost for agitation appears high by reason of the fact that all chemical consumption (cyanide, lead acetate, and lime) is charged in the supplies under this heading. The total cost for these chemicals was \$0.574 per ton, leaving the other supplies charged to agitation amounting to \$0.015 per ton.

The wage scale, on an 8-hr. day, is as follows:

Solution men.....	\$150.00 per month
Solution helpers.....	4.00 per day
Refinery man.....	150.00 per month
Refinery helper.....	4.00 per day
Battery men.....	4.50 per day
Crusher men.....	4.00 per day
Repair men.....	5.00 per day

Having thus outlined the work, costs, and extractions, it may be interesting to analyze the various departments or divisions. A reference

to the flow sheet will show clearly the course of the ore through the mill and the flow of the solutions.

Blake Crusher for Preliminary Work

The ore is delivered to the mill in 4-ton side-dumping cars, each car being weighed before being emptied into the crusher bin. Over this bin are placed rails to prevent any boulders which cannot pass an 18-in. square opening from reaching the bin; from this bin the ore passes over a 2-in. grizzly to a 10 by 16 in. Blake crusher. The discharge from this crusher and the fines which passed through the grizzly are carried by a 16-in. belt conveyor to the battery bins. The total cost of crushing is \$0.081 per ton, of which \$0.045 is for labor; the largest item under supplies is the steel for the crusher, the hardness of a part of the ore causing excessive wear, and extreme fine crushing being necessary.

Stamps, Chilean Mill, and Tube Mill for Secondary and Final Crushing

Stamps, Chilean mill, and tube mill complete the crushing and fine-grinding equipment. The stamps, fed by suspended Challenge feeders, weigh 1,400 lb. each, dropping 96 times per minute through 6½ in., in a narrow mortar. For a crushing capacity averaging 133 tons per day (to compensate for time lost for repairs, replacements, etc., this often reached 180 tons per 24 hr.) screens of 0.120 wire with ¾-in. square opening in the clear were used. Occasionally for several days at a time screens with ½-in. square opening would be used, and less frequently finer screens, depending entirely on the character of the ore.

The shoes are 9 in. in diameter and the dies 9¼ in. in diameter; the stems are of mild steel 4 in. in diameter; the cam shaft, of hammered iron, is 6½/16 in. in diameter. Very little trouble is experienced from broken stems, and the original cam shaft, installed when the mill was built, is still in commission, having been doing duty for over three years, crushing a total of approximately 130,000 tons of ore. The cam-shaft boxes are without caps and not babbitted, the one nearest the cam-shaft pulley being 18 in. long, while the others are 15 in. in length.

Chrome-steel shoes and dies were used throughout the year and the following table gives the consumption of steel per ton of ore, and the cost:

	Pounds
Shoes.....	0.232
Dies.....	0.107
Total.....	0.339
Total cost for steel per ton of ore.....	\$0.018

The ore is crushed in solution in the batteries and no attempt is made at classification between the batteries and the Chilean mill, all the battery discharge being fed direct to the Chilean, which is a Trent 6-ft. mill revolving 30 times per minute.

The screens on the Chilean vary considerably on account of the different tonnage going through the mill, but the standard used most of the time is a "ton-cap" screen No. 693. The drive on this mill is a Morse silent chain drive 5 ft. between centers, which is giving excellent service. Chrome steel is also used in this mill and the average consumption for the year was 0.50 lb. per ton of ore crushed, costing \$0.034 per ton.

Following the Chilean mill and in closed circuit with the tube mill is a Dorr classifier, and the remarkable work it is doing deserves particular mention. Frequently runs of 180 tons per day have been made, all of which passed through this classifier, since it is the only one in the mill, and to this must be added at least 85 tons coming from the tube-mill discharge, making a total of 265 tons per day, and even when overloaded to this extent the only attention it ever receives is a little oiling.

The tube mill is a 5 by 22 ft. Allis-Chalmers, belt driven and making 28 rev. per minute. It has an El Oro lining, and the present liner, which has been in service for one year, still has three months' wear; many of the plates are worn practically smooth except for ridges which were formerly ribs. The cost per ton for lining on the total ore milled was \$0.022 counting on a 12 months' life, but this cost will probably be reduced to \$0.016. The height of the pebble load was kept at the center line both before and after the pebbles dropped out of the lining plates, the lining becoming in effect a smooth lining. No appreciable difference could be detected in the work done nor in the pebble consumption. The consumption of tube-mill pebbles for the year was 3.39 lb., costing \$0.080, per ton of ore ground.

The following screen analyses are the averages of a large number of tests made during the year.

Battery Feed—Everything to Stamps

Mesh	Per Cent.
+ 38	37.6
+ 20	38.6
+ 40	6.6
+ 60	4.4
+100	3.1
+150	2.1
+200	1.2
-200	6.4

Battery Discharge and Chilean Feed

Mesh	Per Cent.
+ 20	46.9
+ 40	14.5
+ 60	8.7
+100	7.5
+150	5.0
+200	1.5
-200	15.9

<i>Chilean Discharge</i>		<i>Tube-Mill Feed from Dorr Classifier</i>	
Mesh	Per Cent.	Mesh	Per Cent.
+ 20	9.7	+100	83.6
+ 40	16.2	+150	7.0
+ 60	14.7	+200	3.0
+100	12.8	-200	6.4
+150	9.4		
+200	6.0		
-200	31.2		

<i>Tube-Mill Discharge</i>		<i>Filler Discharge</i>	
Mesh	Per Cent.	Mesh	Per Cent.
+100	8.7	+100	4.9
+150	10.5	+150	8.7
+200	11.4	+200	10.3
-200	69.4	-200	76.1

The following is a cost analysis for the combined stamping and gerinding, including Dorr classifier, taken from Table II.

	Labor	Supplies	Power	Total
Stamps.....	\$0.081	\$0.031	\$0.073	\$0.185
Chilean.....	0.026	0.053	0.063	0.142
Tube mill.....	0.029	0.119	0.107	0.255
	\$0.136	\$0.203	\$0.243	\$0.582

Agitation and Washing in Pachuca Tanks and Dorr Thickeners

The classifier overflow is pumped to Dorr thickener No. 3, the average specific gravity being 1.110 (approximately 5 tons solution to 1 ton ore). The overflow from this thickener flows to the gold tanks for precipitation and the thickened slime passes to the first of four Pachuca agitators for continuous agitation, each agitator being 15 by 45 ft. The average specific gravity of the slime in the agitators was 1.250 (2 tons of solution to 1 ton of ore).

I wish to call particular attention to the work of this Dorr thickener No. 3. The tank is 24 ft. in diameter and 14 ft. deep, having a surface of 452 sq. ft.; therefore, an average of 133 tons per day allows only 3.4 sq. ft. of surface per ton of ore thickened, and when the tonnage reaches 180 tons per day this is reduced to 2.44 sq. ft. per ton of ore, but undoubtedly the specific gravity of the slime delivered to the agitators on these days was below the average.

Before proceeding to the detailed figures on the agitators it will be interesting to examine into the extraction made to this point. The ore first comes in contact with solution in the batteries and in its passage through the batteries, Chilean mill, classifier, tube mill (a portion of the

ore only) and the Dorr thickener the precious-metal content has been reduced from the average of 0.216 oz. gold and 19.48 oz. silver to 0.051 oz. gold and 5.42 oz. silver, showing an extraction of 76.4 per cent. of the gold, 72.2 per cent. of the silver. I do not know of any other silver ore so amenable to cyanide that in its passage through the grinding machines and one thickener on its way to the agitators an equal percentage can be taken into solution, and I consider this an interesting and unusual silver ore for this reason. It was the intention of the designers that the overflow from this thickener No. 3 should flow to the battery storage tank for battery feed solution, little thinking that it would be the richest solution in the mill, with the single exception of the solutions in the agitators, and that it would have to be precipitated before being used again.

Unless especially mentioned to the contrary all assays quoted in this article are averages of samples taken every day in the year, except for the few occasions when the mill may have been down for 24 hr. continuously; most of the samples were taken by dips or cuts each hour.

Part of the lime supply is added to the ore as it passes from the ore bin to the Challenge feeders, the rest in agitator No. 1, and the same is true of the lead acetate; the cyanide (129 per cent. sodium cyanide) is all added to agitator No. 1 after being dissolved in a small box.

Table III shows the average assay value of the ore as it enters agitator No. 1, its value as it leaves each agitator and as it reaches the filter box, for each month of the year, the average for the year; the percentage extraction and the number of ounces extracted in each step. It will be noted that the extraction in agitator No. 4 was only 0.002 oz. gold and 0.24 oz. silver, and I am satisfied that agitation in this tank could have been discontinued and the same results obtained in Dorr thickeners No. 2 and No. 1 (see flow sheet) before the slime reached the stock tank.

The difference in value of the slime in agitator No. 4 and the filter heads (both washed samples) is 0.003 oz. gold and 0.09 oz. silver, which shows the extraction that took place in the two Dorr thickeners No. 2 and No. 1 and the stock tank in which the slime was washed after agitation. The value of the solution in agitator No. 4 varied from \$5 to \$7 per ton, depending on the value of the ore, and was reduced approximately to \$1 in the stock tank by the washings applied. These two thickeners being each 28 ft. in diameter have 4.63 sq. ft. per ton of slime settled.

The usual titration in agitator No. 1 was about 4.5 KCN and 1.6 CaO. During the cold months sufficient steam was used in agitators No. 1 and No. 2 so that at some time during the 24 hr. their temperature was between 85° and 90° F., the steam being used in pipes in the uplift column.

TABLE III.—*Extraction Table—One Year's Assays*

Ore Heads 0.217 oz. gold, 19.48 oz. silver, value \$15.52

Month	Feed to Agitator No. 1		Discharge from Agitator No. 1		Discharge from Agitator No. 2		Discharge from Agitator No. 3		Discharge from Agitator No. 4		Filter Heads	
	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag	Oz. Au	Oz. Ag
Oct.....	0.062	5.97	0.037	3.69	0.025	2.12	0.016	1.40	0.013	1.17	0.010	1.10
Nov.....	0.056	4.51	0.032	2.75	0.021	1.56	0.016	1.19	0.012	0.95	0.011	0.98
Dec.....	0.060	4.23	0.034	2.71	0.023	1.68	0.017	1.27	0.014	1.02	0.011	0.92
Jan.....	0.066	4.32	0.038	2.42	0.026	1.81	0.020	1.43	0.018	1.27	0.013	1.03
Feb.....	0.058	4.19	0.035	2.79	0.019	1.54	0.015	1.31	0.015	1.08	0.011	0.96
Mar.....	0.055	5.10	0.037	3.21	0.026	2.25	0.020	1.72	0.015	1.37	0.011	1.22
Apr.....	0.070	6.11	0.038	3.50	0.026	2.53	0.019	1.96	0.015	1.63	0.010	1.47
May.....	0.047	6.00	0.026	3.20	0.028	3.48	0.018	2.44	0.013	1.89	0.010	1.88
June.....	0.036	5.71	0.023	3.42	0.015	2.63	0.011	2.05	0.008	1.80	0.007	1.80
July.....	0.030	6.75	0.022	3.64	0.013	2.54	0.008	1.97	0.006	1.76	0.006	1.81
Aug.....	0.035	5.91	0.020	2.70	0.010	1.86	0.006	1.41	0.005	1.30	0.005	1.30
Sept.....	0.033	6.29	0.022	3.36	0.012	2.32	0.007	1.83	0.006	1.75	0.005	1.50
Average.....	0.051	5.42	0.030	3.11	0.020	2.20	0.014	1.66	0.012	1.42	0.009	1.33
Extraction, per cent.....			86.2	83.7	90.8	88.7	93.6	91.5	94.5	92.7	95.9	93.2
Extraction in tanks, per cent.....			9.7	11.5	4.6	5.0	2.8	2.8	0.9	1.2	1.4	0.5
Dissolved in tanks, oz.....			0.021	2.31	0.010	0.91	0.006	0.54	0.002	0.24	0.003	0.09

Dissolved before ore reaches agitator No. 1, ounces, Au 0.166, Ag 14.06. Extraction before ore reaches agitator No. 1, per cent., Au, 75.7, Ag, 72.2.

The specific gravity of the slime in the agitators averaged 1.250 (2 tons solution to 1 ton of ore), consequently, averaging 26 lb. of dry ore per cubic foot. The available area for agitation, after deducting for the cone and to a depth of 2 ft. from the top, is 24,440 cu. ft. in four tanks, giving a total dry weight of ore of 317.2 tons. This gives each tank a capacity of nearly 80 tons and a total period of agitation of 57 hr.

Oliver Filters Used

The filtering installation consists of two 8-ft. and one 14-ft. Oliver filters, all 11 ft. 6 in. diameter. For considerable periods the two 8-ft. filters would handle the slime, but whenever a rush of talc occurred it was found necessary to use the third. Under the former conditions the filters handled 0.23 ton, or 460 lb., of ore per square foot of canvas per day with an 18-in. vacuum. This capacity could be greatly increased by thickening the slime and raising the vacuum. A dry vacuum is used and the solution removed from the receiver between the filters and the dry-vacuum pump by a small triplex pump.

Table II gives the cost of filtering as follows: Labor \$0.079, supplies \$0.040, power \$0.026, total \$0.145. At first glance the cost for labor and supplies seems high for this type of filter, but since they require equally as much attention as the thickeners, zinc boxes, stock tank or boiler, it is only fair that a small proportion of the solution man's time and the helpers' time be charged against them. Also in the way of supplies, only about 20 per cent. of the charge was for supplies for the wheel itself, the rest being for vacuum pumps, triplex pump, slime pump for pumping overflow from filter tanks back to stock tank, etc.

A saving of 0.23 lb. of cyanide was made during this year over the preceding year with practically the same grade of ore, and I believe the greater part, if not all of it, was saved on the filters, due to changing the method of washing the cake, during the last eight months of the year. This change made it possible to wash both ascending and descending sides of the cake with water.

Formerly the method was to wash the ascending side of the cake with barren solution, and the descending side with water when the amount of solution in the mill would permit, or with barren solution when crowded, it being found that a water wash applied at all times would build up too much solution on account of the flow of water into the filter box between the wires during the time the discharged section was moving past the scraper.

Barren solution washes the precious metals out of the cake, but in a silver mill, where all solutions are necessarily high in cyanide, the cyanide loss is considerable. To prevent this loss of cyanide the usual rubber apron was placed against the cake on the ascending side and a

water wash applied, the surface water thus caught being diverted to the slime-discharge launder. On the descending side an iron apron, 18 in. wide, extending the full length of the filter, was placed so that the top edge, resting on supports at the ends of the filter, scraped off just the top of the cake; the lower edge, extending over the edge of the filter box, diverted the part of the cake scraped off (possibly $\frac{1}{16}$ in.) together with the wash to the discharge launder, but did not interfere with the regular blow-off. All that was necessary to accomplish this was to place this apron so that the top edge was just above the blow-off section and at such an angle that the slime and the water wash would flow over it and drop into the discharge launder.

This method permits the use of enough water to keep a constant sheet of water flowing over both the ascending and descending sides of the cake and does not permit any of it to enter the mill solution. It insures a more efficient wash, since it is impossible always to have barren solution for washes free of precious metals. As stated above, the saving of cyanide was 0.23 lb. for the year, and had this method been in use all of the year instead of eight months, it is reasonable to suppose the saving would have been one-half greater.

On account of adding barren solution to the stock tank and decanting same, it was necessary to thicken the slime in the filter boxes. To accomplish this, a small tank was placed below the filters, 2-in. pipes being connected in the filter box at the level at which it was desired to keep the slime and near the center of the drum. This drew off the thin slime and solution on the surface, delivering it to the small tank, from which it was pumped back to the stock tank or to the thickener just above the stock tank. This served the double purpose of keeping thick slime in the filter boxes and preventing an overflow, doing away with the float regulator on the valve controlling the supply.

Before the discharged cake and water have flowed 100 ft. down a steep hill just below the mill the cake and water have become thoroughly mixed, and at about this point a sample is taken each shift for specific gravity, titration, and assay of solution. Thus, a sample for determination of the specific gravity of the discharge stream is taken three times a day. The average of all these for seven months shows a ratio of 1.38 tons of water for each ton of ore. These samples are then allowed to settle and an equal amount of the clear water or solution from each is placed in a sample for titration and assay. The average of all titrations showed 0.62 lb. of cyanide and at the above ratio this gives a loss of 0.85 lb. of cyanide per ton of ore treated. The assays of this solution and the specific gravity are then used to compute the loss in soluble gold and silver, which for seven months was \$0.18 per ton of ore. This regular sampling was in effect during only the last seven months of the year.

Precipitation by Zinc Shavings

Zinc shavings are used for precipitation. Seven 7-compartment boxes are used, each compartment having a V-shaped bottom beneath the screen and emptying into launders which carry the precipitate to canvas-bottomed tanks connected with vacuum for drying. No zinc is used in the first compartment and immediately after a cleanup the boxes contain approximately 444 cu. ft. of zinc shavings. Since 331,750 tons of solution were precipitated during the year, the daily tonnage amounts to 909 tons or 2.05 tons of solution per cubic foot of shavings per 24 hr.

The actual cubical contents of zinc shavings varies from 444 cu. ft. immediately after a cleanup (a cleanup is made every 10 days) to probably two-thirds of that figure just before the next cleanup. This, together with lost time, would considerably increase the tons per cubic foot of shavings, so that if an accurate estimate could be formed it would be found nearer 2.5 tons than the figure given above.

Dividing the total production by the tons precipitated shows a value of \$2.12 precipitated per ton of solution entering the zinc boxes. The zinc-box tails averaged \$0.107 per ton, giving the zinc-box heads a value of \$2.227 per ton and showing an effective precipitation of 95.2 per cent. It is natural to presume that these values varied greatly from day to day and even from hour to hour. Costs for supplies and labor are shown in Table II.

Sheet zinc is purchased in carload lots and cut in the mill with a Johnston zinc lathe. One operator will cut 700 lb. per day. The charge for labor, power, and incidentals, totals \$0.007 per pound for cutting zinc. The shavings are 0.004 in. thick; 9-gauge 36 by 108 in. sheets are used.

Precipitate Refined without Preliminary Acid Treatment

During the year the dry weight of the precipitate taken from the zinc boxes was 98,610 lb., which produced 62,608 lb. of bullion, gross; the bullion averaged 968 fine in gold and silver. Thus the precipitate contained 63.5 per cent. of bullion gross and 61.5 per cent. of gold and silver.

No acid treatment was required, except on one lot of zinc shorts accumulated during the first six months of the year. When the precipitate is taken from the vacuum tanks it is placed in an electric drying pan for drying to approximately 20 per cent. moisture. The flux finally decided upon as the most satisfactory was 7 per cent. each of borax glass, soda, and sand. Melting is done in an oil furnace burning crude oil and using Dixon No. 10 retorts.

General Résumé

The additions necessary to increase the tonnage, enumerated earlier in this article, total less than \$10,000; undoubtedly next year's operations will show a further increase in tonnage and I feel safe in predicting a treatment of 16 tons per stamp per day, counting actual operating time.

Treating this tonnage makes fine preliminary crushing necessary, as will be seen by a glance at the screen test of battery feed. This, however, adds only a small additional cost to crushing, as the total crushing cost was only \$0.074 per ton of ore for labor, supplies, and power. It is doubtful if the extra cost due to crushing to this fineness increased the cost \$0.02 per ton over what it would have cost to crush to the usual size for battery feed. This is a good illustration of the advantage of fine crushing preceding stamps.

The chemical consumption was as follows:

	Per Ton of Ore	Ounces Bullion per Pound
Sodium cyanide.....	1.72	10.56
Zinc.....	1.63	10.19
Lead.....	0.94	19.45

I find myself at a loss to explain satisfactorily the forced use of lead acetate in seemingly excessive quantities during the year's work. The total consumption for the year was 0.94 lb. per ton, but the first six months averaged only 0.74 lb., while the last six months averaged 1.14 lb. At about the time this change in consumption took place assays of the slime discharge and the slime in the agitator indicated that the extraction was falling off. Without waiting to run experimental tests, the usual steps were taken to correct conditions, the temperature of the solutions was raised, alkalinity was raised and later lowered, more cyanide was added, and then, because the lead acetate consumption had been higher than usual, the weight added was cut in two, when suddenly the assays indicated a still further drop in extraction and it was apparent the trouble had been found. Double the amount of lead acetate formerly used was now added and extra lead was added to all tanks following agitator No. 1. The effect was apparent the next day in reduced slime assays all over the mill.

Since that time numerous efforts were made to reduce the lead consumption, but it was found impossible without an excessive increase in the value of the tailings amounting to several times the cost of the lead. In connection with this use of lead acetate and in order to present all available facts I will state that no noticeable difference was apparent in the cyanide and alkaline strengths, nor in the temperatures of the solutions at this time nor later, nor was the value or character of the

ore changed. It is probable one of the two following reasons was to blame for the change:

1. About this time a more completely closed circuit at the filters prevented loss of solution, due to using all water wash instead of a solution wash followed by a water wash. This failure to renew solutions may have been the cause of a higher lead consumption.

2. Sheet zinc was purchased and cut in the mill, at this time, replacing the former practice of purchasing zinc shavings, which may have had some effect upon the solutions.

The perplexing question of how extraction should be figured seems to be a live issue among the fraternity at this time, and without attempting to settle it one way or the other the effect of different methods as applied to the work in this mill can be readily shown.

As explained before, the extraction is estimated on smelter returns and tailings assays. Smelter returns were \$14.46 per ton and tailings \$1.06, giving the ore a value of \$15.52 per ton and a value extraction of 93.2 per cent. This tailings sample, assaying \$1.06, contained a certain amount of solution, since the wash on the filter was not 100 per cent. efficient, and the sample was not washed after being taken. However, a regular daily sample was cut from the stream of slime flowing to the filter each hour of the 24. This sample was washed and assayed, averaging \$0.96. Adding to this the solution loss of \$0.18, shown previously, gives a total value for tailings of \$1.14; this added to the \$14.46 per ton smelter returns gives an ore value of \$15.60, indicating a value extraction of 92.7 per cent., a difference of 0.6 per cent. from the reported extraction. This extraction would certainly be unfair to the mill, however, since it has not been determined accurately what extraction takes place as the slime cake is being washed with water, dropped from the filter and discharged. The slime is thus in contact a short time with what is in reality a weak fresh solution. Most men experienced in this work will agree that some extraction takes place.

On the other hand, suppose extraction were figured on smelter returns and the product showing the lowest assay value in the mill, namely, the filter heads; this method would give \$14.46 smelter returns plus \$0.96 or \$15.42 for a new ore value, and the value extraction on these figures would be 93.8 per cent. This is manifestly an unfair extraction figure, because the treatment was not yet completed when the filter-heads sample was taken.

It is the general practice to compute extractions by using a tailings sample secured somewhere about the filter after the cake has been washed, either before or after it has been dropped. In the work at this mill it is taken after the cake has been dropped. Therefore, the slime contains certain solution values forced back into the cake by the compressed air used for discharging. I maintain that this is a fairer sample

than one taken before the cake is dropped from this type of filter, or any filter discharged by allowing water or air to enter the vacuum chamber. It must be recognized that there are several viewpoints from which to approach the question of calculating extraction, and that one should be chosen which seems to be the fairest to all concerned.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West, 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

Oil, Gas, and Water Content of Dakota Sand in Canada and United States

BY L. G. HUNTLEY,* PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

Introduction

IN view of the recent advance made in the knowledge of the nature and conditions accompanying the occurrence of oil and gas, and of the recent activity in drilling in Wyoming, Montana, and western Canada, a discussion of the probabilities of obtaining production in such a reservoir as the Dakota sand in general, and in particular districts, is thought by the author to be timely and likely to result in more intelligent operating in certain parts of the territory underlain by this sand. In Canada particularly it is thought that these general considerations have equal if not more significance in a consideration of the oil and gas possibilities of the formations than have the various local conditions.

Age and Nature of Deposition of the Dakota Sand

The Dakota is distinctive in that it is known to form a generally continuous sheet of sand underlying the greater part of Alberta, Saskatchewan, Manitoba, North and South Dakota, Montana, and large areas in Nebraska, Colorado, and Wyoming. It does not seem to have been laid down in more or less isolated lenses, as has been the case with most oil-sand horizons.

Except in northern Alberta, the Dakota sand contains fresh-water fauna, having been deposited very uniformly over these wide areas in shallow water with prevailing strong currents. In northern Alberta marine and brackish water conditions prevailed. In this district it forms the basal member of the Cretaceous sediments, directly overlying beds of Devonian limestone along a great unconformity in the Athabasca River valley and eastward. In southern and western Alberta it is underlain unconformably by the Kootenay shales (Lower Cretaceous), and by Jurassic and Carboniferous rocks. In the United States in South Dakota

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it was in places deposited upon Cambrian quartzites, and in other places directly upon the granite. In the State of Dakota, and in the outcrops about the Black Hills uplift, the Dakota sand is underlain by from 200 to 350 ft. of sandy shales comprising the Fuson formation, the Lakota sand, and the Morrison shales, all of Lower Cretaceous age.

Westward from the Black Hills these formations thin out, and in the Big Horn Mountains are represented by the "Cloverly" formation. It is believed the "Cloverly" includes the Dakota, the Fuson formation, and the Lakota, lying conformably on the Morrison shales (Jurassic or Lower Cretaceous).

The principal point to be brought out is that in most of this area, except as mentioned above, the Dakota was laid over an unconformity at the base of the Upper Cretaceous measures.

In the Athabasca River country of northern Alberta, the Dakota is overlain by the black carbonaceous Clearwater shales (Benton?), and by the Grand Rapids sands (Niobrara?). Some authorities include these in the Dakota group, while others consider them to be of Niobrara age. Overlying the Grand Rapids sandstone series are about 900 ft. of black carbonaceous LaBiche shales. In southern and western Alberta and Montana the Dakota is overlain by the Benton, Niobrara, and the Claggett or Lower Dark shales. These are all of marine origin, and are unbroken by any continuous sand horizons except a few irregular lenses of limited lateral extent. In southern Montana and Wyoming the Dakota is overlain by shales of the same general character, but included in them are a number of locally continuous sand bodies, some of which have proved productive of oil and gas. Among these are the Shannon sand of the Pierre, and the Wall Creek sand of the Benton, in the Salt Creek and Powder River pools in Wyoming; and the sandy Aspen formation in the lower part of the Benton shales, which contains most of the oil in the Spring Valley district in Uinta County, Wyoming.

Lithological Character of the Dakota

A typical section of the Dakota sand consists of a lower basal member of conglomerate followed by beds of coarse porous brown sand with some clay. The upper portion of the horizon is often split into lenticular sand bodies alternating with shale, and frequently indicates a return to brackish and marine conditions. In this upper portion are frequently found locally continuous beds of limestone. Only along the western shore line of the Dakota horizon at its outcrop in the foot hills and in the Moose Mountain district in western Alberta and British Columbia, is there a pronounced change from the general section given. Here the basal bed of conglomerate and coarse sand persists, but this is overlain by shales and quartzitic sandstones to a thickness of almost 1,000 ft.

This remarkable uniformity of outcrop indicates a continuity which is further attested by many wells drilled throughout this area, as well as by the generally artesian character of the Dakota sand throughout North and South Dakota and central Canada. Its character places it in the category of what Roswell H. Johnson describes as a "sheet" sand, including such sands as the æolian St. Peter's sandstone, and the Sylvania sand of Ohio. That is to say, it is an actually continuous sheet of porous sand with no extensive shale bodies interrupting its continuity, as opposed to a horizon such as the Bartlesville sand in Oklahoma, which is a name given to a lateral series of more or less connected sand bodies or lenses, occurring at a certain position in the columnar section.

As stated previously, the economic evidence of this sheet nature of the Dakota sand is its having the general character of an artesian sand reservoir underlying the great synclinal basin west of its outcrop in Dakota and Manitoba. As such, it is one of the valuable natural resources of these districts.

The Dakota thins out in central and western Wyoming, and shows considerable cross-bedding along its outcrop in the Big Horn Mountains. In parts of this area it consists of a series of thin sandstones alternating with beds of shale. On this western edge it is considered as being included in the Cloverly group of sandstones, which probably includes the Fuson and Lakota formations of Lower Cretaceous age. (G. H. Eldridge: *Bulletin No. 119, U. S. Geological Survey*, p. 22.)

Extent of the Dakota Outcrop in Canada

The outcrop of the Dakota in the Moose Mountain district has already been mentioned. Besides the outcrops along the Athabasca River, few exposures have been noticed in northern Saskatchewan and Manitoba. (Wyatt Malcolm: *Memoir 29-E, Geological Survey of Canada*.)

1. On Carrot River, 4 miles above the west line of Indian Reserve.
2. Along the south shore of Lake Wapawekka, to the east of Lac LaRonge.
3. On Beaver River, just above the mouth of Dore River, banks of white sand 90 feet high.
4. South shore of Isle de la Crosse.
5. In various water wells drilled back of outcrop in southern and eastern Manitoba.

By referring to Fig. 1, it will be seen that from any point along this eastern and northern outcrop seepage conditions are down-dip to the great central Moosejaw synclorium. As the Dakota outcrops in a region of lakes and muskeg swamps southeast from the height of land, its being a reservoir filled with fresh water throughout wide areas is explained, drainage conditions having kept its exposure more or less saturated with entering surface water since Cretaceous time.

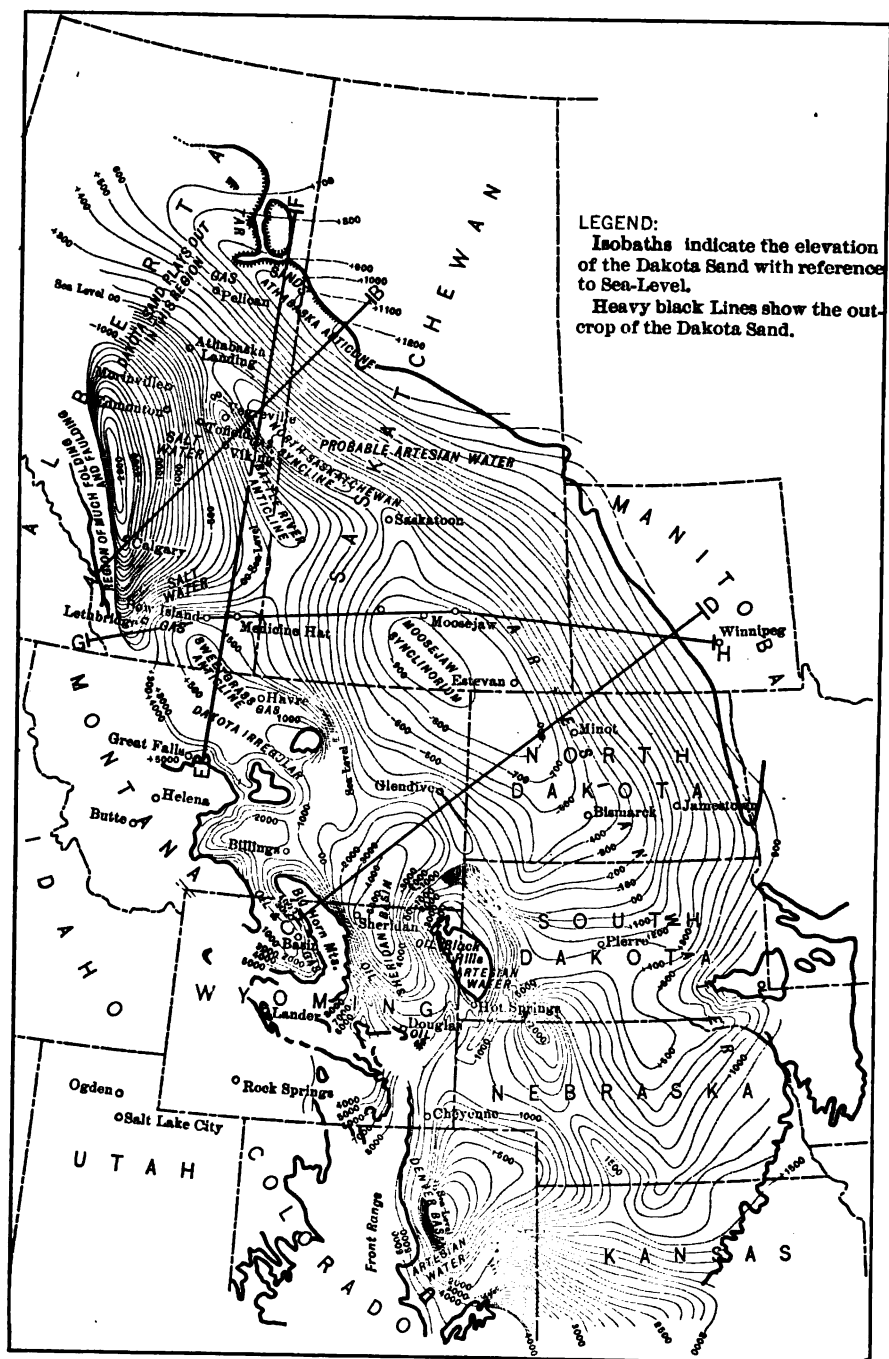
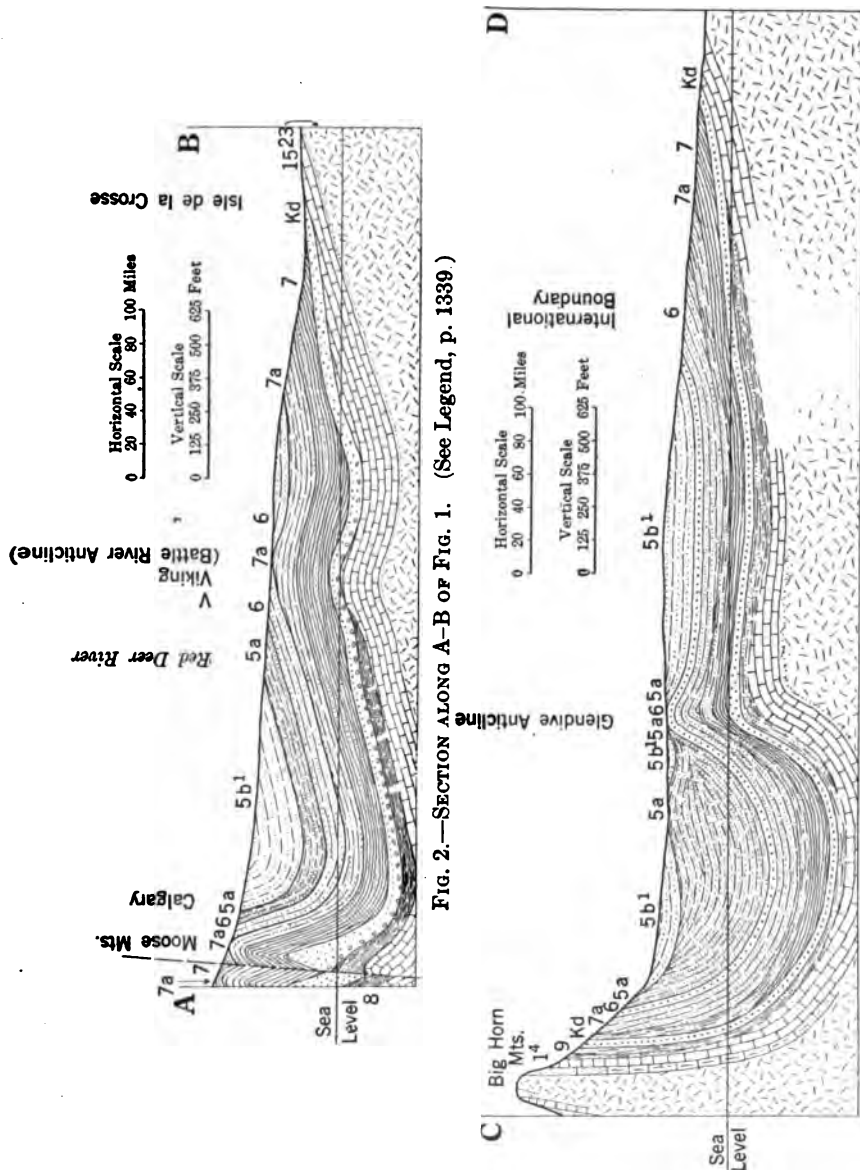


FIG. 1.—Sketch Map showing Generalized Structure of the Dakota Sand in the United States and Canada, with Relation to the Oil, Gas, and Water Reservoirs in that Sand Body.

The Dakota sand plays out in the region between the Athabasca and the Peace Rivers. On the latter river it is apparently represented by the Loon River shales at the base of the Cretaceous measures. (Mc-



Connell: *Memoir, Canada Geological Survey of Canada*). It was also not recognized as a sand in the deep well drilled at Morinville, 18 miles north of Edmonton, Alberta.



FIG. 4.—SECTION ALONG E-F OF FIG. 1.

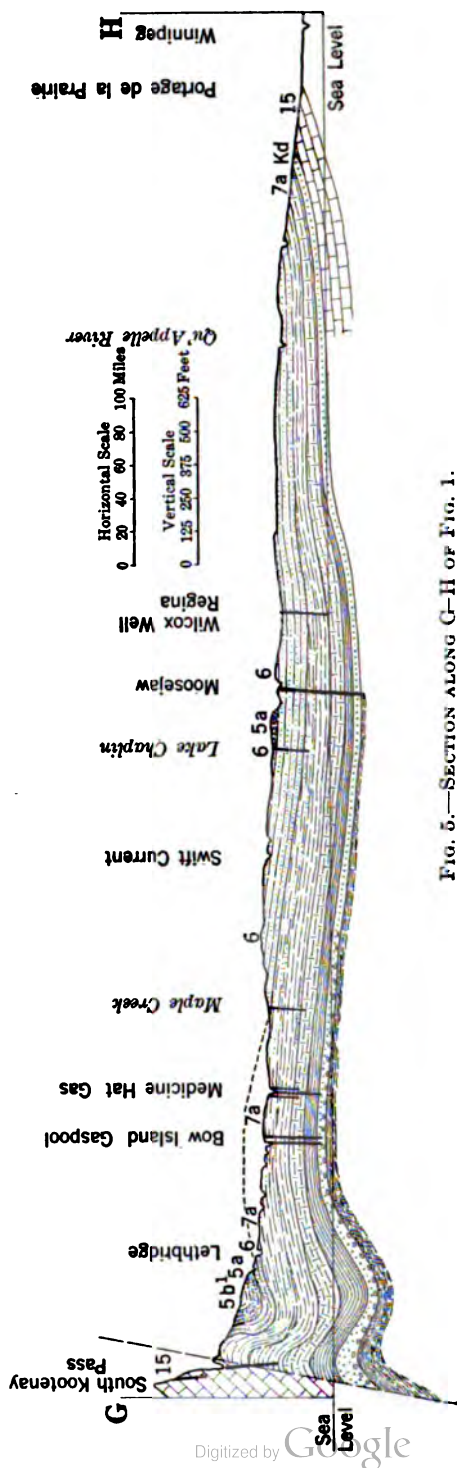


FIG. 5.—SECTION ALONG G-H OF FIG. 1.

Legend—(applies to sections, Figs. 2 to 5)

Lower Tertiary.	5b1, Laramie—Paskapoo Series.	Fresh-water sands, clays and shales.
Upper Cretaceous.	5a, Laramie—Edmonton Series (coal bearing).	Sand and shales.
Upper Cretaceous.	6, Bearpaw (Pierre-Foxhill).	Gray-brown shales, sand shells.
		Sand and shale in upper portion, black shale below.
Upper Cretaceous.	7a, { Belly River and Lower Dark shales.	
	{ Niobrara (cardiom).	Sand lenses and dark shales.
	{ Benton.	Black and gray shales
Upper Cretaceous.	Kd, Dakota sand.	Soft, porous sand (250 to 950 ft.), conglomeratic at base.
Lower Cretaceous.	8, Kootenay shales (coal-bearing).	
Devonian.	15, Devonian.	Limestones, shale and salt or gypsum.
Cambrian.	18, Cambrian.	Reddish sand and shales.
Archean.	23, Laurentian.	Granite.

Tar Sands, and Production of Oil from the Dakota Sand

Where the Dakota outcrops along the Athabasca River, from Crooked Rapids to Fort McKay, and in the tributary streams on either side, it is found to be saturated with inspissated petroleum or "tar," from its top to its bottom. This tar-saturated sand continues up the Clearwater River to the eastward, and it is also reported in the vicinity of Buffalo Lake across the line in Saskatchewan. A considerable "lake" or seepage of this "tar" is also reported in the country between the Athabasca and the Peace rivers just back from the probable outcrop (Fig. 1), and probably exudes from the Dakota sand a few feet below the surface. This seepage is reported to have been increasing in size in recent years.

A small amount of dry sulphurous "tar" is also found coming out of the old government well at Pelican Rapids along with the gas.

Petroleum has been produced from the Dakota horizon in the Powder River pool and the Big Horn Basin in Wyoming; and it is quite possible that deeper drilling in the Salt Creek pool will prove its existence at this horizon on that dome. Small seepages have been reported from the Dakota outcrop on the western slope of the Black Hills uplift, and a little heavy oil has been produced from the Moorcroft pool. (C. V. H. Barnett: *Bulletin* 581, *U. S. Geological Survey*.)

Analyses of Oil Produced from the Dakota Sand Horizon

The following analyses of oils produced from the Dakota horizon are given, as indicating what may be expected in different localities:

A sample of oil from the Moorcroft pool, Crook County, Wyoming, according to David T. Day, had a specific gravity higher than 0.9000. It is a heavy oil suitable for the manufacture of lubricants. Paraffin base.

Specific gravity.....	0.9198
Baumé, degrees.....	22.2
Color.....	dark green
Begins to boil.....	130° C.
Gasoline, per cent.....	0.5
Kerosene (0.858 specific gravity), per cent.....	24.0
Residuum (0.94 specific gravity), per cent.....	76.0
Paraffin wax, per cent.....	2.5

Average analysis of three samples of oil from the Powder River pool, analyzed by David T. Day:

Specific gravity.....	0.9123
Baumé, degrees.....	23.46
Begins to boil.....	213° C.
150° to 300° (0.8528 specific gravity).....	17 cc.
Residuum (lubricating oil, paraffin and tar, 0.9295 specific gravity).....	81.2 cc.
Total cc.....	99.1
Water, per cent.....	0.9
Sulphur, per cent.....	0.395
Asphaltum, per cent.....	2.39
Unsaturated hydrocarbons:	
Crude, per cent.....	30.7
150° to 300° C., per cent.....	7.3

Analysis of oil from Well No. 1 of the Athabasca Oils, Ltd., 9 miles north of Fort McKay, Alberta. Analyzed by the Mines Branch of the Canadian Department of Mines, Ottawa:

	Per Cent.
Water, approximate.....	5
Gasoline and naphtha, 0° to 150°.....	10
Kerosene, etc., 150° to 300°.....	15
Residue.....	70

Analysis of oil from Well No. 2 of the Athabasca Oils, Ltd., 9 miles north of Fort McKay, Alberta. By Falkenburg & Laucks, Vancouver, B. C.:

Specific gravity.....	1.02
Gasoline and naphtha, 0° to 150°, per cent.....	7.1
Kerosene, etc., 150° to 300°, per cent.....	60.1
Base fixed at 300° C., asphaltum, per cent.....	32.8

Analysis of oil from the Wyoming Oil & Development Co. well on Sec. 8, T. 32, R. 73 W., in the Douglas field, Converse County, Wyoming. Made in the U. S. Geological Survey laboratory under the direction of Dr. Day:

Specific gravity.....	0.8439 (35.9 Bé.)
Color.....	olive green
Begins to boil.....	80° C.
0° to 150° (0.7205 specific gravity), per cent.....	8.0
150° to 300° (0.7928 specific gravity), per cent.....	38.5
Residuum (0.9340 specific gravity), per cent.....	53.5
Paraffin, per cent.....	2.0
Asphaltum.....	none
Sulphur, per cent.....	0.2
Water.....	none

Analysis of oil from the Union Pacific well near Spring Valley, Uinta County, Wyoming. Analysis made by Wilbur C. Knight:

Specific gravity of crude.....	0.81 (44° Bé.)
Flash point.....	70° F.
Temperature given off—Degrees C.	Specific Gravity
77 to 130	0.7230 Oil at 15° C.
130 to 170	0.7540 Oil at 15° C.
170 to 200	0.7800 Oil at 15° C.
200 to 259	0.8040 Oil at 15° C.
259 to 292	0.8190 Oil at 15° C.
292 to 320	0.8340 Oil at 15° C.
320 to 350	0.8470 Oil at 18° C.
350 to 370	0.8580 Oil at 22° C.
370 to 380	0.8640 Oil at 22° C.
380 to 400	0.8880 Oil at 22° C.

The above analysis represents about the following composition:

Gasoline and lighter, per cent.....	20 to 30
Kerosene, per cent.....	30 to 40
Paraffin, per cent.....	10 to 20
Lubricating (residue).	

Gas in the Dakota Sand

A strong flow of very "dry" gas was found in the Dakota sand by a well drilled by the Canadian Geological Survey at Pelican Rapids on the Athabasca River in 1897-98. This flow has been continuous until the present time. The gas has a strong odor of sulphur, but no petroliferous smell whatever, nor has any condensation been noticed near the well head even in the coldest winters. It is evidently a very "dry" gas consisting principally of methane.

A considerable gas pool has been developed in the vicinity of Bow Island in southern Alberta; 17 or 18 wells have been drilled in the pool,

and are producing gas from the Dakota sand. Their total estimated capacity is something more than 75,000,000 cu. ft. per day. The gas has the following analysis:

	Per Cent.
CO ₂	0.0
CO.....	0.0
Oxygen.....	0.10
Heavy hydrocarbons.....	1.80
CH ₄ (methane).....	86.70
Hydrogen.....	5.40
Nitrogen.....	6.00

Recently a well drilled by the Edmonton Ad Club 5 miles north of Viking, Alberta, on the author's location, obtained a considerable flow of gas from what may be the Dakota sand. Although this gas is reported dry and odorless, the discharge from the well showed upon analysis a small petroleum content. The well was evidently drilled near the edge of the salt-water zone, as after blowing open for a week or 10 days, it began to make considerable brine. Analysis showed the gas to be high in methane, and of about 1,000 B.t.u. per cubic foot.

Gas and oil are produced from the Cloverly formation in the Big Horn Basin in Wyoming.

Salt Water in the Dakota Sand

No salt or fresh water has made its appearance in the Pelican government well as yet. However, as stated in a preceding paragraph, the Dakota sand may be saturated with salt water up the dip almost to the well where gas was struck at Viking, or approximately 100 ft. above sea level. The gas sand in this well may, however, be a sand body above the Dakota.

A well drilled at Brooks in southern Alberta found a strong flow of salt water in the Dakota at an elevation of 460 ft. below sea level. The Dakota sand in the Bow Island gas field is found at approximately 500 ft. above sea level.

Fresh Water in the Dakota Sand

This sand is an important artesian reservoir in South Dakota, eastern North Dakota, southern Manitoba, in the vicinity of the Black Hills uplift, in eastern Colorado, and in parts of Kansas and Nebraska. This fresh-water zone extends in Canada at least as far west as Moosejaw in Saskatchewan, and probably includes most of the central synclinal basin and the flanks of the North Saskatchewan syncline. The flanks of the Calgary Basin are probably saturated with salt water in the Dakota sand west of Bow Island and Viking. The reasons for this assumption

will be discussed later. Fresh water is associated with the oil in the Powder River pool and in the Moorcroft pool, on either side of the Sheridan Basin.

Gas is also found at places in South Dakota associated with fresh water in artesian wells.

Structure

The extensive faulting of the formations along the eastern edge of the Rockies is flanked by a great structural basin or trough, whose axis passes through Calgary (Fig. 1), at which point the Dakota would be found at approximately 2,300 ft. below sea level (5,500 ft. below the surface). East and north of this great structural basin occur three broad anticlinal folds:

a. The southernmost of these crosses the International Boundary in the vicinity of the Sweet Grass Hills in Montana (Sweet Grass anticline).

b. The central fold extends in a southwest direction from near Vegreville, crossing the Saskatchewan line near the 52d parallel (Battle River anticline).

c. The northeastern outcrop of the Cretaceous formations in Canada represents in a general way the axis of the northernmost of these three folds, and the one the attitude of whose sides shows the most gentle dips. This has been called by the author the Athabasca anticline.

It may be said that the prevailing dips encountered in the formations east of the axis of the Calgary Basin are remarkably gentle and uniform, and show practically no local irregularities or distortion due to violent secondary folding or torsional stresses. This has undoubtedly been an important factor (in connection with the "sheet" character of the Dakota sand) in the accumulation and redistribution of oil and gas in Alberta and western Canada generally. It will have to be taken into consideration in all prospecting for oil and gas in this region.

Southeast of these three great arches lies a broad, deep, structural basin (Moosejaw synclinalorium), which occupies a large portion of North and South Dakota, northeastern Montana and southern Saskatchewan. The outcrop of the Dakota sand on the northeastern flank of this basin is nowhere more than 900 ft. above sea level. At the base of the Black Hills uplift it is about 1,000 ft. Its outcrop in the Black Hills is from 3,000 to 5,000 ft. On the flanks of the Big Horn Mountains its outcrop is from 4,000 to 6,000 ft., which is approximately true in the Lander district in Wyoming. It rises with the general uplift southward to more than 7,500 ft. above sea level at its outcrop in Colorado.

In a basin between any two of these uplifts the best chances for prospecting for oil in the Dakota would seem to be in the local domes and anticlines which may be found along the side of the basin where the Dakota outcrops at the higher elevation. This is believed to be the

case for the reason that there will be a constant movement of water from the upper side, issuing in springs along the lower side outcrop; while water entering from the upper side may do so locally, and, being under slight head, in seeking the line of least resistance to flow will avoid strong folds or domes. On the other hand, the strong hydraulic head existing along the side of the basin where the lower outcrops occur will tend to keep the sand saturated with encroaching water under pressure, which will usually issue at the low points of a vertically irregular or "scalloped" outcrop. However, the higher areas on such an irregular outcrop will be exposed to inward-seeping surface waters, which will tend to a movement of all water, and any possible oil which had risen to such points, to an exit at lower points on the outcrop where the hydraulic head from the opposite side of the basin is effective.

Thus, not only would "structure" be relatively less effective in retaining oil on the lower side of such a basin, but migrating oil carried with the water movement would be forced to the edge and lost by erosion and evaporation. On the higher side of the basin the tendency would be for oil to be carried down into the sand, and retained in irregularities in the structure.

The small quantities of oil found associated with fresh water along the lower side of such a basin will probably be of heavy gravity.

While the foregoing considerations apply to the Dakota sand, yet the best chances for developing production on the sides and edges of such a basin as that between the Black Hills and the Big Horn Mountains will probably be in overlying sands of a more lenticular shape, or at least having an outcrop on only one side of the basin. This is the case regarding the Shannon and Wall Creek sands in the Salt Creek pool.

Pressures

The rock pressure at the well located at Pelican on the Athabasca River is reported by H. L. Williams to be 225 lb. per square inch. As the difference in elevation between the top of the sand at this point and at its nearest outcrop at Boiler Rapids is very close to 500 ft., it will be seen that this corresponds very closely to that which might be expected on the assumption of Orton's hydrostatic theory. In this district the Dakota sheet sand under consideration offers almost ideal conditions for the application of this theory of underground pressures.

The outcrops of the Dakota along the Athabasca are also the nearest (and the lowest) to the well at Viking on the Battle River anticline, where what is supposed to be the Dakota was struck at approximately sea level. This gives a theoretical head of 1,060 ft., or an equivalent of 460 lb. pressure, which was the initial pressure encountered in this well.

When one comes to the Bow Island field, with initial pressures of

800 lb., the situation is complicated by the lack of knowledge as to seepage conditions in the Sweet Grass Hills, and also the density of the salt water in this district, which Orton apparently neglects. However, there is no important deviation from the pressures which might be expected in the Bow Island district on the assumption of a head of water from the Sweet Grass Hills. The hydraulic gradient would tend to dissipate a great part of this pressure (Darton: *Professional Paper No. 56, U. S. Geological Survey*) before reaching the Viking well, as outcrops of the Dakota are open on several sides of the Bow Island pool. This is based on the assumption of the gas-bearing formation at these two points being identical. The log of the lower part of the Viking well was badly kept, and the author has seen no results of the fossil evidence found. In case the gas-bearing sand is an upper lens, the above agreement still holds true, as in the flat muskeg country in the Athabasca Valley both formations outcrop on the same plane and at about the same elevation above sea level.

Geological History of the Dakota Sand

1. As has been shown, the Dakota sand was deposited over a wide area in shallow fresh water with prevailing strong currents. It was probably in part æolian.

2. There then followed the gradual sinking below sea level of the whole basin, but deepest in the central and western portion, as the Cretaceous sea covered the Great Plains area. Then followed the deposition of black marine shale deposits, with a few local sand bodies as brackish or fresh-water conditions might prevail locally for short periods. These deposits were very general over wide areas, as evidenced by the extent over which we find the Benton, Niobrara, and Claggett or Lower Dark shales, and the Clearwater shales of the Athabasca River.

3. As shown by King (19th *Annual Report U. S. Geological Survey*) and Roswell H. Johnson (The Role and Fate of Connate Waters, *Bulletin No. 98*, February, 1915):

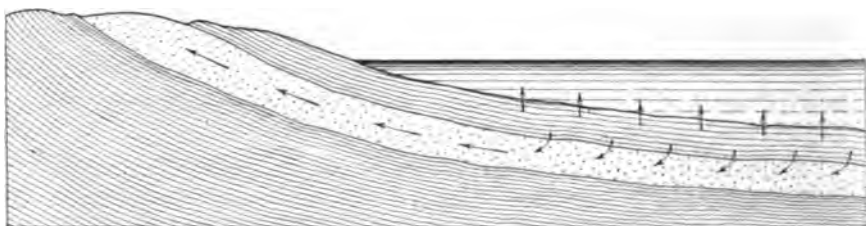
"As any particular sand body becomes weighted by a heavier and heavier overburden, the result of increasing deposition, a part of the large percentage of water in the uncompacted sand and mud is forced out. The water percentage of freshly settled mud is vastly higher than that of the resultant shales, or even of very porous sand. If beds of less compressible material meet or underlie the shales in question, there will frequently be a lateral motion along such beds to some point where upward movement will be resumed."

This is shown by King's diagram (Fig. 6).

On the assumption that most oils are formed in shales rich in organic matter in close proximity to sands which now contain these products, the source of oil (or gas) in the Dakota sand must have been the overlying

marine shales which are even yet so rich in carbonaceous matter. The unconformity upon which the Dakota was laid in the district in which it has been proved to be so rich in tar, that of the Athabasca region, presumably eliminates the lower formations in that district as a source of such hydrocarbons. The relative scarcity of organic life in a fresh-water sand such as the Dakota and the underlying fresh-water Comanchean (or Jurassic) makes doubtful in most districts the possibility of the formation of any great amount of hydrocarbons within the sand body itself, or within those beds immediately below it. It is more likely that oil found in such underlying sands had its origin in contiguous shale beds, or even possibly in shales overlying the Dakota itself, having migrated in some such manner as shown by King's experiments cited above.

In general, it is believed that no oil which has its origin in a lower formation will be found in a bed which has been deposited unconformably upon older and already consolidated rock deposits, unless after the later deposition some tectonic change takes place.



After King, 19th Annual Report, U. S. Geological Survey, part 2, p. 80

FIG. 6.—DIAGRAMMATIC SECTION SHOWING THE FLOW OF CONNATE WATER, OIL, AND GAS, DUE TO CONSOLIDATION OF SEDIMENTS.

4. As demonstrated by King, a portion of the water of deposition of shales overlying a coarse sand body is squeezed downward into the region of greater porosity, and passes laterally toward any outcrop where it may again have a chance to resume its upward movement. This movement of the connate water would therefore carry with it a considerable portion of any oil and gas which had been formed, segregation taking place selectively during its passage through regions of varying porosity.

During Cretaceous times the land surface to the northeast was being very gradually raised, offering an outcrop toward which movement of the liquid content of the Dakota took place. As pointed out by Roswell H. Johnson:

"Oil, gas, and water have extremely slight capacity for gravitational sorting while in a state of rest, but when moving the gravitational sorting is readily accomplished. As oil, gas, and water pass a body of larger pores, the gas, owing to its lack of capillary attraction, is retained in the larger pores." (*Rôle and Fate of Connate Water in Oil and Gas Sands, Bulletin No. 98, February, 1915.*)

Much oil was therefore probably carried with the water to the outcrop and from there lost by evaporation and erosion, leaving a relatively higher proportion of gas remaining in the sand along with the residual water. Where the escape and erosion were not so rapid as on the northeastern front, oil would be forced by this movement out into the periphery of the basin and there retained, the deeper central portions of the Dakota sheet being occupied entirely by water (possibly salt) and some gas.

In regions of cross-bedding and lenticular deposition of the sand, of course such movement and gravitational sorting would be complicated by other barriers.

5. After such distribution of the fluid content of the shales and sands had been effected, in late Cretaceous time came the Black Hills uplift, the Big Horn uplift, and the upward tilting of the whole southwestern portion of the sedimentary blanket underlain by the Dakota sheet sand, in which already a considerable concentration of hydrocarbons had taken place. This tilting had an enormous effect on the hydraulic conditions in the Dakota sheet. The whole northeastern front had by this time been exposed by erosion, and fresh water had probably entered through seepage from this outcrop, already tending to force any residual oil body toward the lowest point of the "V" between the higher fresh-water sealed outcrop and the northern edge of the sand lens between the Athabasca and the Peace Rivers. Then, when the great southern and western uplift took place and a large proportion of the reservoir was raised above the general level, and folding formed the Calgary and Moosejaw synclines and their corresponding folds, this hydraulic head caused a great flood of water through the Dakota northward, probably carrying with it some of the remaining oil. The synclinal basins were already saturated, so to again establish equilibrium, the excess water was forced out by way of the outcrops to the north and east. By this scouring action in the sand, according to the principle of "selective" segregation, the dry gas clung to the more porous portions at the crest of the structural arches, the water spilling across the structural saddles or divides.

The remaining portion of the oil carried by the water was then either (a) forced to the outcrop and lost; or (b) forced into closely overlying sand lenses or crackled shale; or (c) forced out into the region where the Dakota is still under cover and of less uniform porosity—that is to say, along the boundary of this sand body, where more or less isolated lenses of sand offered catchment for such oil. Such lenses as these, however, probably already contained oil and water which had been left relatively undisturbed by such movements.

The remains of the action first mentioned are seen in the extent of the northern Athabasca tar sands, and the frequent fragmentary remains of the tar-saturated sand found with the glacial drift on the plains of

Alberta south of the outcrop of the Dakota. It may be mentioned that the lightest oil found in this tar-sand district has a specific gravity of 1.02, as shown in a previous paragraph.

The probabilities of the second action do not increase the promise of finding large oil pools or reservoirs in such shales or minor splits of the Dakota overlying the main sand body, but rather will be indicated by paraffin shales or by thick, viscous water-washed tar occurring near the top of the sand, such as that found in the hole drilled at Pelican Rapids, in the upper part of the gas-bearing Dakota.

The possibilities of the third action mentioned above offer the chance of developing important oil pools in the Dakota sand in certain areas whose geological limits are already approximately known, lying west and south of the Athabasca River. In the southern part of the Dakota area, in the United States, in parts of Montana, Wyoming, Nebraska, and Colorado, where the area underlain by this sand is much disturbed and folded by mountain forming, there are of course many unprospected areas for oil and gas in the Dakota. These represent pools of oil and gas cut off by varying porosities, by intrusions and accompanying folding, from reaching the outcrops of the sand, as in the Big Horn Basin, and the Salt Creek and Powder River pools.

In this southwestern part of the Dakota Basin the conditions of deposition were more irregular, and there were probably a number of islands in this part of the Cretaceous sea. These conditions, combined with later folding, had a tendency to limit the movement of fluids in this sand to local areas and lenses, which have already resulted in the development of good oil and gas pools. The relative time of folding is a factor which is often neglected in considering the oil and gas possibilities of a formation in certain districts.

With reference to the Moose Mountain district in Alberta, and the series of sharp folds south and west of Calgary, it will be noticed that there is a considerable thickening of the Dakota series along the mountains, to approximately 1,000 ft., composed principally of quartzose sandstone and shale beds, with the basal conglomerate bed from 30 to 60 ft. in thickness.

As the great Cretaceous mediterranean sea deposited layer after layer of marine ooze and mud as an overburden to the porous Dakota sand, this former shore line of the Dakota Basin constituted a very open outcrop for the upward movement of circulating fluids continually being forced into this bed from overlying compacting shales (Fig. 6). Moving laterally along the sand, this water suddenly entered the thickened section at the upward-bent shore line. Such conditions probably established a sharp pressure gradient within a short distance, which may account for the deposition by ascending waters of the siliceous cement which caused the quartzitic condition of the Dakota along this western outcrop.

While such conditions may have sealed an outcrop locally, yet in general the thickness of the sand along this belt presumably offered too open a condition to permit of any considerable accumulation of oil back of a generally sealed edge. If this hypothesis is correct, the chances of finding oil in commercial quantities in the Dakota in that region are greatly lessened.

Summary

1. In general, on account of the very uniform character and thickness of the Dakota sand over wide areas, it is not believed to be a prospect horizon of the first class for the development of oil production, except in certain limited districts.

2. In Canada it is believed that the most favorable areas for the development of oil production in the Dakota sand will be found in the region underlain by this sand where it begins to play out and become discontinuous and lenticular in its nature. This is the case in the oil pools of Wyoming producing from the Dakota sand, and points to certain parts of Montana as worthy of prospecting.

3. While not a prospect of the first order, the testing of the Battle River anticline for oil in the Dakota is distinctly worth while. This anticline is, for the foregoing reasons, more favorable for gas in this sand; and oil should be looked for, if at all, in the more laterally restricted sands occurring in the Benton or Niobrara, or higher.

4. The chances for the development of oil production in the formations lying below the Dakota in western Canada, are not of the first class from a commercial standpoint.

5. Where structural and other, conditions are favorable for the concentration of oil, the relatively more lenticular sand bodies above the Dakota, particularly in the Benton and Niobrara, are more promising for the development of production than is the Dakota sand itself; except in such regions as mentioned in the preceding paragraphs.

*Logs of Alberta and Montana Wells**Pelican Oil & Gas Co.'s Well No. 1, Pelican Rapids, Northern Alberta.*

(Dec. 3, 1912. Well Head 1,300 ft. A.T. approximately)

Formation	Top	Bottom	Formation	Top	Bottom
Blue and yellow shale...	1	66	Dark gray shale, sandy..	767	843½
White and gray shale			Sandy shale.....	843½	872
(water).....	66	82	Sandy shale.....	872	882
Blue shale.....	82	200	Coarse rock mixed with		
Blue and brown shale..	200	235	heavy oil.....	882	887
Brown shale.....	235	285	Shale and sand.....	887	898
Gray-brown shale.....	285	331	Hard rock.....	898	903
Sand rock (hard).....	331	352	Lime carrying oil.....	903	997
Shale.....	352	365	Limestone.....	997	1,051
Sand rock.....	365	425	Hard flinty shell (strong		
Shale.....	425	507	flow gas under shell)...	1,051	1,053½
Brown shell (hard).....	507	509½	Limestone.....	1,053½	1,158
Gray shale.....	509½	538	Hard lime shell.....	1,158	1,159
Shell.....	538	540	Limestone.....	1,159	1,192
Sandstone.....	540	546	Hard shell (gypsum)....	1,192	1,197
Shale.....	546	575	Blue shale and gypsum..	1,197	1,293
Hard shell.....	575	581	Hard lime shell.....	1,293	1,296
Gray shale, streaks			Lime rock.....	1,296	1,538
sandstone.....	581	644	Lime, shale, and lime		
Strong flow gas.....	625		rock.....	1,538	1,560
Gray shale (gas).....	644	653	Gray shale and lime (gas)	1,560	1,700
Gray shale (soft and			Limestone.....	1,700	1,784
cement-like).....	653	666	Hard shell.....	1,784	1,790
Sand rock.....	666	671	Lime rock (shale streaks)	1,790	1,875
Gray shale.....	671	688	Hard shell.....	1,875	1,879
Hard brown shell.....	688	689	Layers of limestone and		
Dark gray shale.....	689	740	shale.....	1,879	2,040
Hard shell.....	740	741	Strong flow of gas.....	2,040	
Dark gray shale.....	741	766	Limestone and shale, in-		
Hard shell.....	766	767	terstratified.....	2,040	2,069

Morinville Well No. 1, drilled by H. L. Williams, near Egg Lake, Alberta.

Formation	Top	Bottom	Formation	Top	Bottom
Surface drift with clay boulders.....	0	250	Gravel (salt water).....	2,900	2,902
Sand rock.....	250	260	Greenish shale, like "dobe" shale.....	2,902	2,940
Blue and brown shale, thin layers of sandstone.....	260	440	Sandrock (oil), flow gas underneath.....	2,940	3,052
Sand rock with gas.....	440	465	Hard shell with iron....	3,052	3,064
Blue and brown shale with one or two thin layers of sandstone....	465	1,410	Greenish shale, like "dobe" shale.....	3,064	3,100
Dark blue shale with heavy oil seep.....	1,410	1,415	Blue shale with thin layers of sandstone....	3,100	3,200
Blue shale, dark blue to light blue and green....	1,415	2,450	Greenish shale, very sticky (dobe).....	3,200	3,260
Hard ironstone shell....	2,450	2,456	Hard ironstone shell....	3,260	3,262
Blue and gray shale with gas.....	2,456	2,900	Blue sandy shale with oil seepages.....	3,262	3,310
			Blue shale, interstratified with hard lime shells...	3,310	3,340

From 3,040 to present depth, more or less oil, increasing. Shale from 3,040 very hard, with pyrites, but slacked on exposure.

Fort McMurray—Great Northern Exploration Co. Well No. 2.

(Finished August, 1912.)

Formation	Top	Bottom	Formation	Top	Bottom
Surface (soil).....	0	17	Lime.....	502	562
Limestone.....	17	77	Shale.....	562	592
Shale.....	77	92	Lime.....	592	604
Lime.....	92	152	Salt (salt-water).....	604	704
Shale.....	152	192	Limestone.....	704	779
Soft shale.....	192	197	Salt.....	779	869
Lime.....	197	237	Lime.....	869	999
Shale.....	237	242	Shale.....	999	1,059
Lime.....	242	362	Brown sandstone.....	1,059	1,119
Shale.....	362	382	Brown "Medina" rock	1,119	1,139
Lime.....	382	462	Red rock, streaked, hard	1,139	1,405
Shale.....	462	502			

Viking Gas Well, 5 miles due north of Viking, Alberta.

Formation	Top	Bottom	Formation	Top	Bottom
Yellow clay.....	0	30	Sandy white shale.....	465	485
Blue clay.....	30	40	Brown shale.....	485	515
Gray sandy clay (little water).....	40	50	Grey shale (gas est. 50,- 000 ft.).....	515	530
Blue clay.....	50	75	Brown shale (gravelly and caving, gas).....	530	535
Shale.....	75	85	Brown sand (gas).....	535	585
Shale.....	85	100	Brown shale.....	585	655
Sandy shale.....	100	120	Gray sandy shale (water)	655	680
Loose sandy shale (water)	120	140	Brown shale.....	680	690
Blue slate.....	140	165	Water sand (slightly salt)	690	700
Light brown shale.....	165	170	Gray sand (salt water)	700	740
Gray shale.....	170	190	Blue shale.....	740	890
Gray sandy shale.....	190	210	Shell.....	890	895
Gray shale.....	210	230	Brown shale.....	895	1,210
Light brown shale.....	230	243	Hard lime shell.....	1,210	1,215
Coal.....	243	245	Blue shale.....	1,215	1,400
Dark brown shale.....	245	252	Brown shale.....	1,400	1,605
Gray sandstone.....	252	272	Sandy gray shale.....	1,605	1,610
Gray shale.....	272	280	Sandy shell.....	1,610	1,615
Brown shale.....	280	340	Brown shale.....	1,615	1,700
Sandy shale shell (water)	340	345	Bastard sand and shale	1,720	1,770
Brown shale.....	345	360	Brown shale, (gas in quantity).....	1,770	2,180
Gray shale.....	360	370	Blue shale.....	2,180	2,335
Brown shale.....	370	390	Hard capping, boulder formation (gas in quan- tity).....	2,335	2,340
Hard shell.....	390	395			
Brown shale.....	395	403			
Sandy shell (gas and water).....	403	405			
Gray shale.....	405	465			

Montana Oil & Development Co. Well, near Shelby, Mont.

(Drilling started July 16, 1912)

Formation	Top	Bottom	Formation	Top	Bottom
Black shale.....	40	160	Light shale.....	1,150	1,195
Lime shell.....	160	162	Hard shell.....	1,195	1,200
Black shale.....	162	315	Hard sand.....	1,200	1,223
Water and gas sand.....	315	320	Shell.....	1,223	1,230
Hard shell.....		320	Sand.....	1,230	1,300
Shale.....	320	420	Light shale.....	1,300	1,360
Gas sand.....	420	430	Sand.....	1,360	1,390
Sandy shale.....	430	460	Hard shell.....	1,390	1,400
Hard shale.....	460	470	Light shale.....	1,400	1,460
Black shale.....	470	650	Hard sand.....	1,460	1,500
Gray sandy shale.....	650	720	Hard shell.....	1,500	1,510
Black sand.....	720	730	Hard sand.....	1,510	1,550
Sandy shale.....	730	770	Yellow shale.....	1,550	1,600
Light shale.....	770	850	Gritty sand.....	1,600	1,650
Sandy shale.....	850	950	Hard shell.....	1,650	1,655
Black shale.....	950	1,045	Black shale.....	1,655	1,675
Gray sand.....	1,045	1,065	Hard shell.....	1,675	1,680
Gas sand (some gas)....	1,065	1,070	Lime rock.....	1,680	1,730
Black shale.....	1,070	1,100	Black shale.....	1,730	1,755
Light shale.....	1,100	1,115	Bottom of hole—		
Red rock.....	1,115	1,150	abandoned.		

Vegreville Well No. 1, Vegreville, Alberta

Formation	Top	Bottom	Formation	Top	Bottom
Surface.....	0	15	Bottom sand.....		1,362
Blue mud.....	15	272	Brown shale.....	1,362	1,440
Brown shale.....	272	325	Blue mud.....	1,440	1,558
Gas.....	328		Little gas.....	1,563	
Shell 2 ft. thick.....	330	386	Brown shale.....	1,563	1,740
Water.....	386		Blue mud.....	1,740	1,868
Sand.....	386	410	Sand (gas 1,870).....	1,868	1,872
Blue mud.....	410	515	Blue mud.....	1,872	1,880
Gas.....	520		Brown sand.....	1,890	1,920
Blue mud.....	520	1,000.3	Blue mud.....	1,920	2,000
Sand.....	1,000.3	1,356	Bottom of hole.		
Gas.....	1,360				

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Correlation and Geological Structure of the Alberta Oil Fields*

BY D. B. DOWLING, † B. AP. SC., F.R.S.C., OTTAWA, ONT., CANADA

(San Francisco Meeting, September, 1915)

THE interest which has been aroused in prospecting for oil in the foot hills of southern Alberta, and in the oil possibilities of the known gas fields situated in the less-disturbed areas, called for a much closer examination of the structure, thickness, and composition of the underlying rocks of the region than had hitherto been made. The areal geology of the larger part of the great plains was outlined by Dawson, McConnell, and Tyrrell, between 1881 and 1885. The foothill area was not critically examined at that time, owing to the time which would have been required for its proper study and the difficulty that was found in recognizing in the foothills the divisions which had been adopted in the mapping of the formations of the plains. This was due in great measure to the paucity of exposures in continuous sections of the lower divisions of the Upper Cretaceous. Since the pioneer work on the plains was published, the beds which form continuations south into Montana have been critically examined. The Canadian beds arch over the end of a flat anticline which rises to the south; and two sections in Montana, one on each side of the anticline, help to explain some of the difficulties previously encountered in the mapping of the measures found in the foothills. The index map, Fig. 1, shows the area studied, and its relation to southeastern Alberta, and to Montana.

Problems of Correlation

The principal difficulty experienced in correlation was in connection with the measures known as the Montana group. The beds of this group did not seem to agree in the two Montana sections. They showed the same differences to some extent as are encountered in comparing the foothill measures with those of the plains.

The Montana group in Dakota and Manitoba consists of a series of shales of marine origin. In the eastern Montana section two sandstone members appear, leaving a shale member at the top and another

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† Geologist, Geological Survey, Department of Mines, Ottawa, Canada.

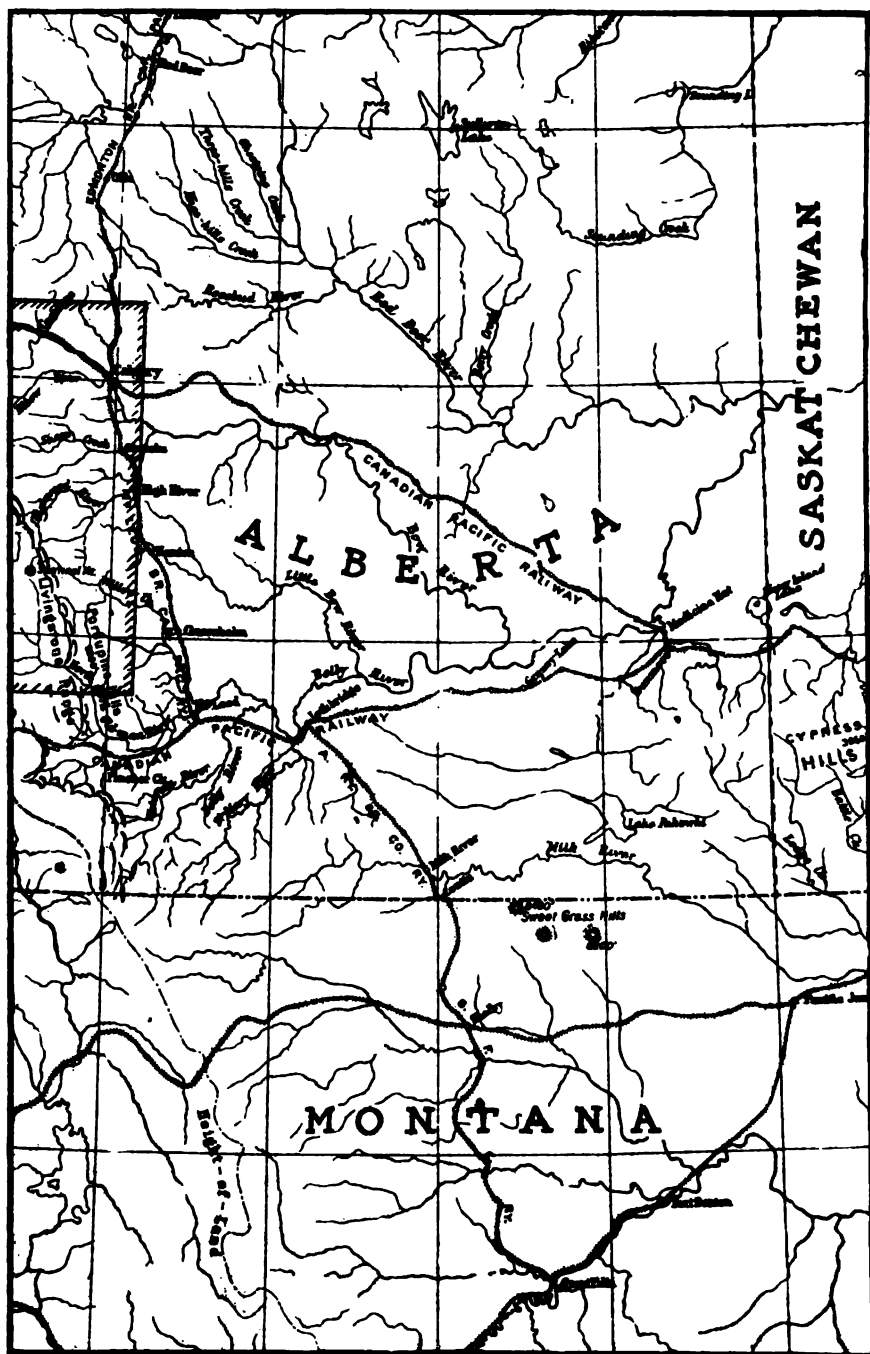


FIG. 1.—INDEX MAP OF PORTION OF MONTANA AND ALBERTA.

near the bottom. In the section in western Montana the lower shale is not found and the upper shale is reduced in thickness (Fig. 2).

The eastern and western sections have been correlated by tracing them to the International Boundary and making a connection by means of the formations on the Canadian side.



FIG. 2.—IDEALIZED WEST-EAST SECTION OF CRETACEOUS SEDIMENTS NEAR THE CLOSE OF THE MARINE INVASION.

The nomenclature adopted for the divisions of the Upper Cretaceous in various parts of western Canada and in the United States is given in the accompanying table.

These formations are found in the foothills, and it is evident that in any section that will be examined in the Montana foothills west of Cutbank we may expect the probable absence of the upper shale member and the appearance of additional sandstone members in the shales of the Benton beneath. Sections to the east will probably show a gradual diminution of the sandstone members. As the area under consideration lies to the north similar variations in the deposits no doubt occur and a few notes on the sections there observed are here introduced.

Dakota Group

The Dakota sands, which contain gas at various points, heavy asphalt oils on the Athabaska and some light paraffin oils in the vicinity of Calgary, are probably represented in the Sweet Grass Hills by one of several sandstone beds which apparently lie at the base of the Benton. To the west these sandstone beds thicken as the mountains are approached; and in the Blairmore section on Crowsnest River the basal beds lying above the coal measures of the Kootenay are upward of 2,000 ft. in thickness. To the north they are less than 1,000 ft. thick in the foothills west of Calgary, and decrease to 200 ft. on the Athabaska at some distance from the mountains.

Colorado Group

The Colorado group, mainly shales with an occasional sandy member, underlies probably the whole of the plains and is found inside the mountains. Its thickness in the Sweet Grass Hills is estimated to be 850 ft., and in the vicinity of the mountains at Blairmore 2,700 ft. In the foothills it may well be more than 2,000 ft.

	Southern Alberta		Central Montana	Manitoba	South Dakota
	Western Montana	West Forks of Milk River	East near Pakowki Coulee		
Montana Group	Bearpaw shales (<i>Marine</i>)	Foxhills Pierre	Bearpaw shales (<i>Marine</i>)	Odanah shales (<i>Marine</i>)	Foxhills
	Two Medicine formation (<i>Fresh and brackish</i>)	Belly River formation	"Pale" and "Yellow" beds Sandstones and clays (<i>Fresh and brackish</i>)	Judith River formation (<i>Mainly fresh water</i>)	Pierre
			Shale at Forks	Millwood shales (<i>Marine</i>)	(<i>Marine</i>)
	Virgelle Sandstone		"Castellated" rocks of Milk River	Eagle sandstone	
Colorado Group	Benton shales (<i>Marine</i>)	Lower Dark shales (<i>Marine</i>)		Benton shales (<i>Marine</i>)	Niobrara shales (<i>Marine</i>) Benton shales (<i>Marine</i>)
Dakota Group		Exposures of sandstones in the Sweet Grass Hills, and also reached by drilling in Alberta.		Exposures in Sweet Grass Hills	

Montana Group

The character of the deposits of the Montana group, as shown in the table above, indicates that the sea margin changed several times but remained during this period within the area now occupied by the foothills and the plains, and that in tracing these beds eastward a diminution in the thickness of the sandstones and an increase in the shales may be looked for. Northward the problem of the variations in the deposits is being studied, and a few points that have been determined may be mentioned here.

It is fairly well established in the section between Medicine Hat and the mountains that the sandy beds at the base are very thin at Medicine Hat and in the foothills resemble the "castellated rocks" of the southern section. Northward they evidently become thin again, as they do not appear in the Athabaska section and may be represented in the mountains by the Brazeau formation or the Dunvegan sandstones of Peace River. The marine shales, called "Claggett" in the eastern Montana section, lying between the sandstone members, are probably represented at Medicine Hat by a much thicker shale series. In the foothills near the Crowsnest River this shale series has thinned out and cannot be recognized. It is probably present in the sections north of the Highwood River, increasing in thickness to the northeast, while the sandy beds of the underlying lower part of the Belly River series decrease materially and ultimately thin out. This shale would then, in the absence of lower sandstones, represent the series overlying the Colorado shales of the Athabaska River and of the northern foothills. The upper sandstone and clay member of the Belly River formation is thicker in the west than in the east and the section west of the Porcupine Hills resembles that in western Montana in the inclusion of all the Belly River rocks in one thick sandstone member.

The Bearpaw shale, the upper part of the Pierre, is a marine deposit that overlies the Belly River beds. It is exposed over large areas in southern Alberta and in a somewhat diminished form is found in the foothills to the south. Its thickness is apparently maintained westward; but the marine portion is confined to the central part of the section. The beds at the base, in the vicinity of Lethbridge and even farther east, contain coal-bearing zones, which in places have valuable coal deposits. In the foothills, coal seams are found both at the top and the bottom of these shales, and it is suspected that north of Highwood River the marine deposits of the formation are very thin if they are present at all; so that the division between the Edmonton sandstones and the "pale beds" of the Belly River formation may be marked farther north by merely a coal-bearing zone or by transition beds, instead of marine shales. Some indication of this is given in the character of the rocks in

the section on the Saskatchewan River east of Edmonton. The sandstone and clays of the Edmonton formation do not end abruptly, but merge by degrees downward into a shale series. The shales overlies sandstones which may possibly represent the top of the Belly River series, there composed of a thin sandstone member. The shales beneath are no doubt directly continuous with the Pierre of the east.

Foothill Sections

The geologist working in the foothills of central Alberta must therefore be prepared to find, not the well-marked formations so far discussed as found in other places, but, for the higher beds, a great series of sandstones, ranging in age from the earlier deposits of the Montana group, up to Tertiary deposits. There may be but traces of the marine beds which are found in the sections on the plains to the east. It may be noted also that the Benton formation is well developed, and forms the great shale series of the foothills, but that north of a line drawn northwest from the forks of the Highwood River, shales belonging to the lower part of the Montana group cap the Benton.

Structure

The structure of the outer portion of the foothills has been partly mapped and a comprehensive view of its general character may be gained from the accompanying sketch and sections, Figs. 3 and 4. It will be seen from them that there was an uplift of the lower measures toward the mountains, accompanied by profound fracturing throughout the disturbed zone. Since the lines of fracture penetrate below the beds containing the possible oil supply the fault blocks have necessarily a limited oil-drainage area and do not afford promising ground for wells. Our field of study has been limited, therefore, to the eastern edge of the broken country in the hope of finding anticlines in close connection with the less-disturbed beds of the Alberta syncline which lies to the east. One of these on Sheep Creek is being thoroughly prospected and one well is now producing a small quantity of very light oil. Another in Township 23 west of Elbow River has yielded heavier oil.

In the country to the east of this broken area and the syncline indicated in Figs. 3 and 4, the beds are so slightly flexed that they seem at any one point to be almost horizontal. They are as a rule less consolidated than the beds near the mountains, and the rivers are deeply trenched. This river erosion is accompanied in nearly every case by a series of land slips, extending back for some distance from the banks. This already has been interpreted as faulting by several "experts" and an intricate structure showing anticlines and faults has been pictured providing many "oil companies" with attractive prospectuses. There

is a wide anticline, however, in southern Alberta, between the outer foothills and the Cypress Hills at the eastern boundary of the province, which extends from northern Montana well into Alberta. This had already been the subject of investigation for a possible natural-gas reservoir; and the wells at Bow Island which supply Lethbridge and Calgary are located on it. Attention has again been called to it by the discovery of slight signs of oil in springs on the slopes of the Sweet Grass Hills in Montana, and several drilling rigs have been placed in the valley of Milk River and even at the Boundary line, on the flanks of the above-named hills. The borings in this vicinity will probably penetrate the sandstones of the lower part of the Belly River series, and also the Benton, before reaching the Dakota, from which there seems to be some chance that gas at least will be obtained. The thickness of the Cretaceous measures is here smaller than in the foothills, and very deep wells will not be necessary to test the ground.

A flat anticlinal structure is also indicated by the outcrop of the Belly River rocks in the eastern part of Alberta. This anticline runs in a northwest direction and is crossed by several stream valleys, notably that of the Battle River. The Grand Trunk Pacific Railway crosses the Battle River near the axis of the anticline. A well, sunk for gas near the railway but to the west of the center of the anticline, struck a small gas reservoir at a depth of 2,340 ft.

Development

Oil seepages have been known for many years in the mountains along the International Boundary east of the Flathead Valley. Several companies bored wells at the outer edge of the mountains, and about six years ago there was some excitement over the discovery of oil in a well near the Waterton lakes. The difficulty of getting machinery to this region and the probability of the area being limited prevented extensive prospecting. The finding of oil last year in an easily accessible area of less broken country at the outer edge of the foothills at once attracted the attention of the speculative element of the population; and many companies were formed and oil leases applied for. The discovery well is situated on an anticline of Benton shales, flanked on both sides by sandstone ridges cut through by the valleys of three streams. Since the sandstone at the crown of the anticline has been removed by denudation, the direction of the anticlinal axis is marked by a series of transverse valleys eroded in the shales. These depressions afford favorable locations for derricks; and 11 wells are now being bored. In the country to the west of this anticline many other drillings are being made, so that in the portion of Alberta shown in Figs. 1 and 3 there were during 1914 about 36 separate points of attack, mainly in the foothill belt. Two wells

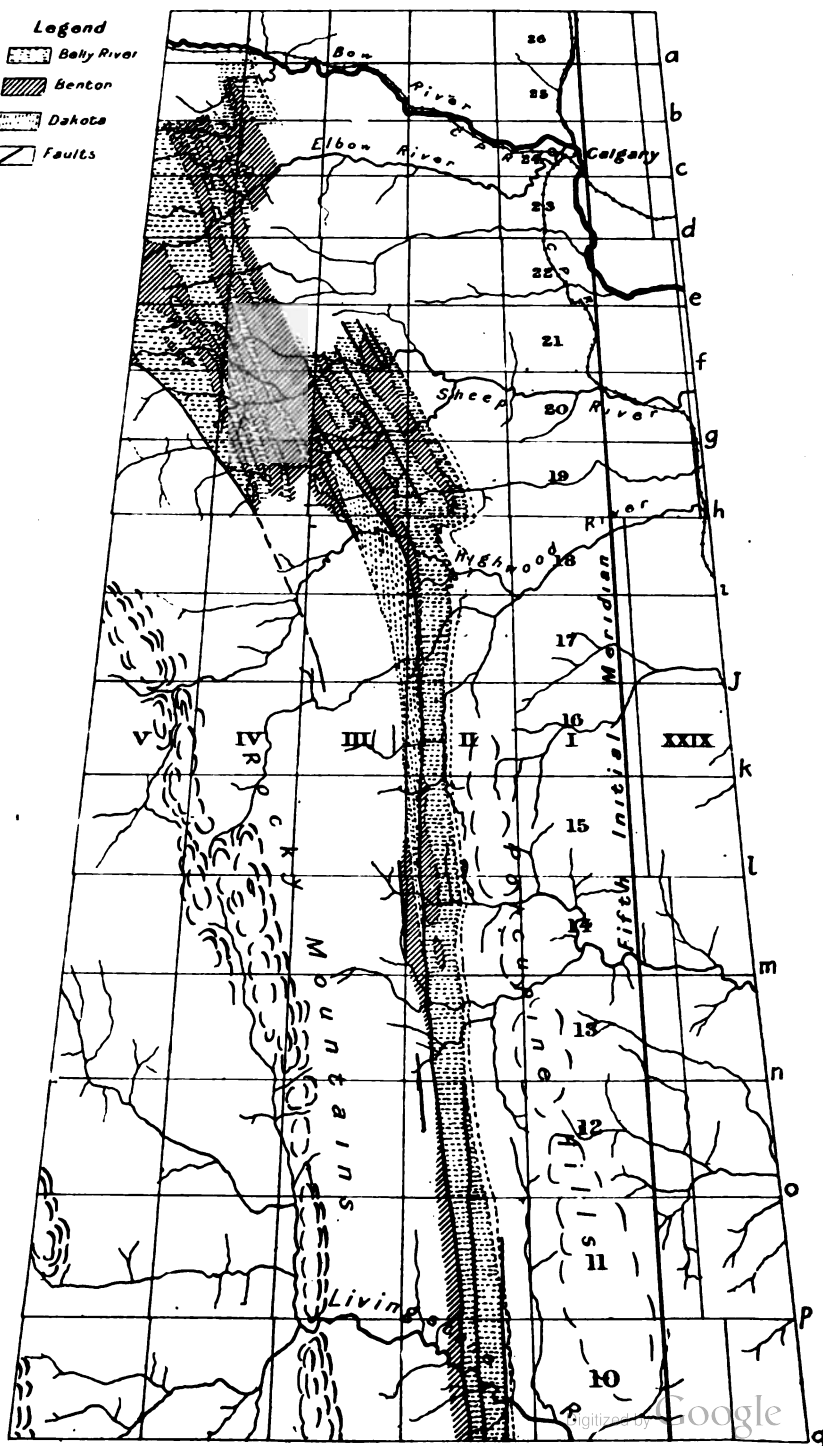


FIG. 3.—PERSPECTIVE DIAGRAM, FOOTHILLS OF SOUTHERN ALBERTA.

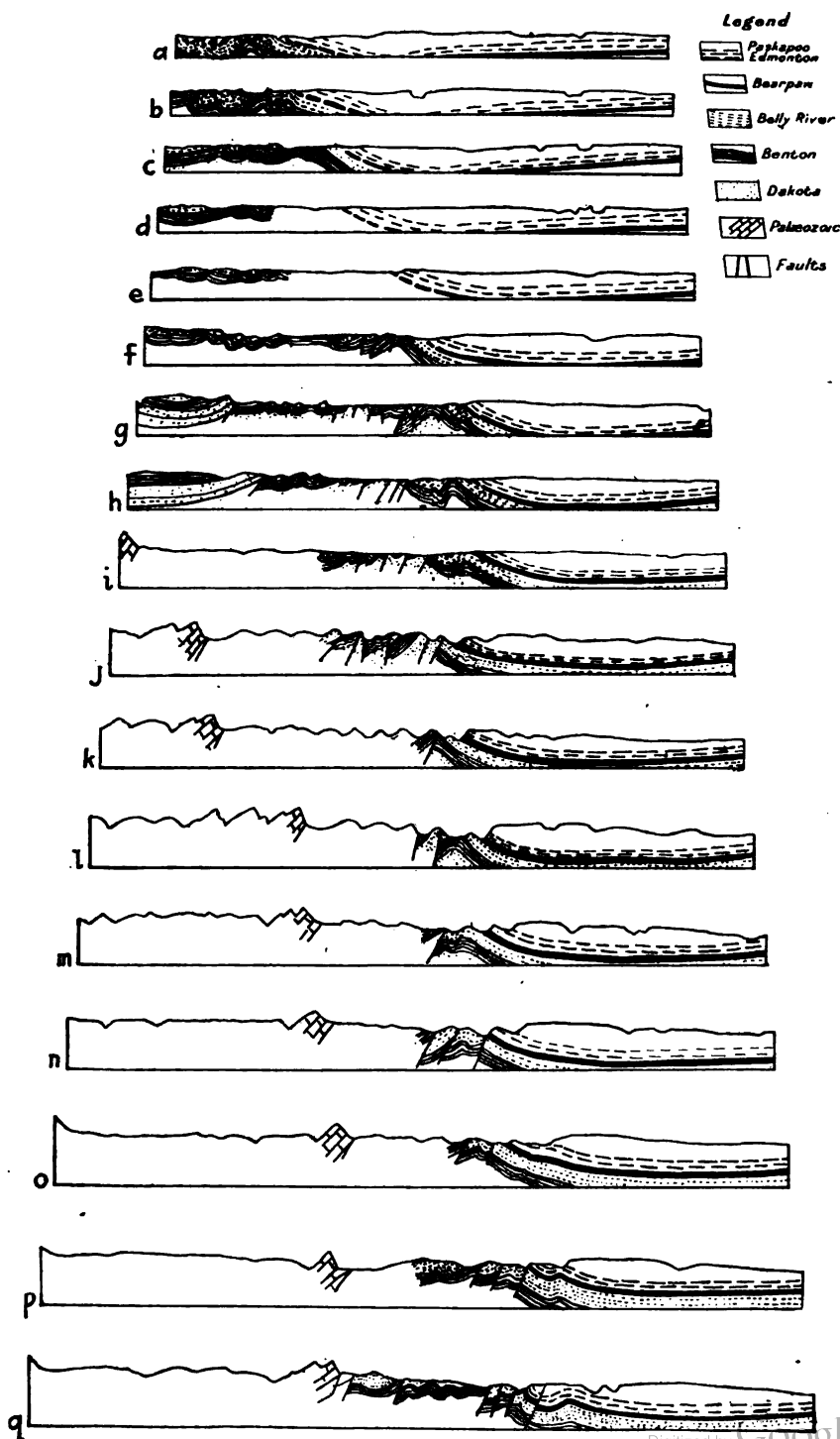


FIG. 4.—STRUCTURE SECTIONS.

have reached depths of more than 3,000 ft. without success. Seven, including the discovery well, are over 2,000 ft. deep. Fourteen are over 1,000 ft. deep and thirteen others have reached smaller depths.

Three companies are boring in the Milk River Valley and four in the foothills north of Bow River. In a few cases it may be considered that the ground has been found to be barren of productive reservoirs; but in the majority of cases the mechanical difficulties have been so great, owing to the depth to the prospective oil sands, that no positive result has been reached. In some cases the wells have been badly located from the viewpoint of structure.

In the discovery well, light gasoline oil and a heavy gas flow was found at 1,550 ft., in sandy beds in the lower Benton. At 2,700 ft. another flow of gas and oil was found in the Dakota or in sands of about that horizon. This oil was also light in specific gravity (about 55° Baumé) and was accompanied by a heavy flow of gas which has been shown by experiment to produce a light gasoline on condensation.

It is claimed that showings of oil have been got in several wells in the vicinity.

A discovery of oil 40° Baumé in the well of the Moose Mountain Oil Co. was announced on Nov. 24, 1914. The well has since been shot and a yield of 25 bbl. per day is claimed. The oil is dark brown and shows a greenish color by reflected light. It was struck at a depth of 1,690 ft. in the top beds of the Dakota.

In March, 1915, two wells near the discovery well reported oil. The Heron-Elder well, on the western limb of the anticline, reached the top of the Dakota at 2,746 ft. Oil came into the well at 2,774 ft. and rose about 2,000 ft. The oil is dark in color and probably heavier than that from the discovery well. About a mile south and near the crest of the anticline the Western Pacific well reached the top of the Dakota at 2,150 ft. and report gives about 300 ft. of oil in the well accompanied by a strong gas pressure.

Location of Oil Discoveries

The locations of these wells may be found on Fig. 3. The townships are numbered from south to north and the ranges west from the initial meridian.

Moose Mountain well, Tp. 23, R. V, west of 5th meridian, about the center of the northern boundary of the township.

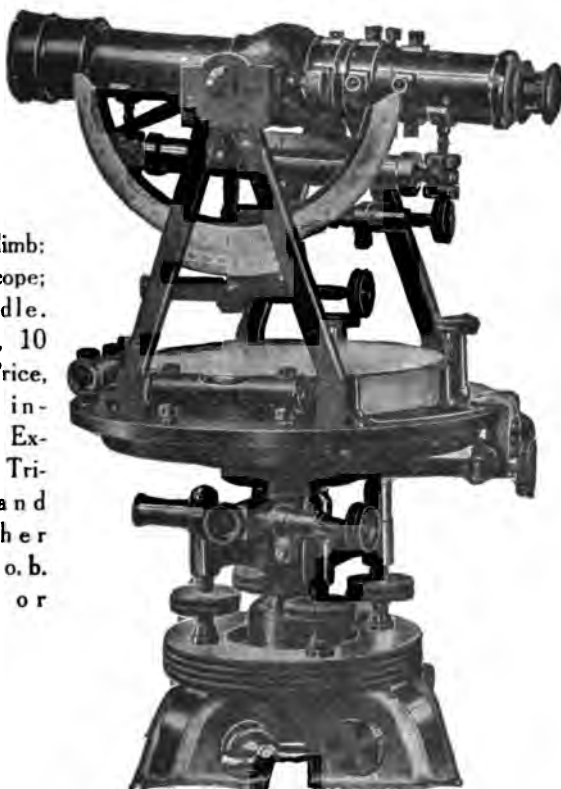
Discovery well, southwest corner of Tp. 20, R. II, west of 5th meridian.

Heron-Elder well, southeast corner of Tp. 20, R. III, west of 5th meridian.

Western Pacific well, northwest corner of Tp. 19, R. II, west of 5th meridian.

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Every undertaking is limited to that point where tools and machinery for its successful performance and completion may be purchased or manufactured. No work can be planned without having in mind such limitation, and the technical journals may be relied upon to furnish descriptions, both editorially and through the advertising pages, of methods and plant used in successfully performing various tasks.

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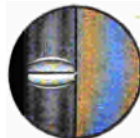
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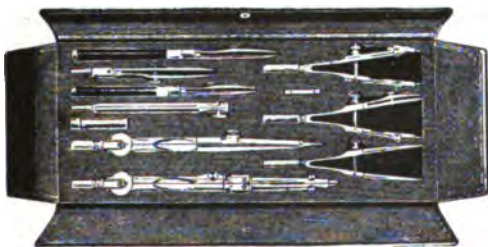


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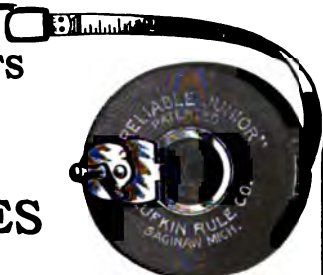
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Shifting Coal with a Loader

The illustration herewith shows one of the uses to which a light wagon loader is put at the immense coal-storage plant of the Lehigh Valley Coal Sales Co., at South Plainfield, N. J.

At this plant greater quantities of coal are discharged into storage than can be removed by the reclaiming machines. As a result there usually remains a large crescent-shaped pile, which must be moved by hand into the area covered by the pivoted reclaimers.

The expense of such hand-shoveling is considerable, and yard superintendents have been seeking a mechanical means of shifting, which would materially reduce this expense. A solution has evidently been found by the company above mentioned by the use of a light wagon loader manufactured by the Hudson Machinery Co., Lawyers Building, Passaic, N. J.



This machine picks up the coal as it runs from the wooden retaining wall, from which one of the boards has been removed, performing this operation at the rate of 40 to 50 tons per hour and practically without the aid of an attendant.

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Before the invention of any device for the accurate recording of weight of material transported on belt or bucket conveyors, cable railways, or overhead transporters there was an element of uncertainty which made for inefficiency, and which no doubt resulted in loss of profits.

A machine is now on the market known as the Merrick Weightometer. Its manufacturers claim for it an accurate record of weight of all raw material entering into or leaving the plant, which record serves to check shippers' weights, and furnishes a daily report on material received, and amount taken from stock. This permits of definite knowledge of material on hand without any guesswork or laborious calculations.

It is interesting to consider the value of such an apparatus in detecting the particular points of underproduction, whether due to faulty or insufficient machinery or careless and indifferent operators.

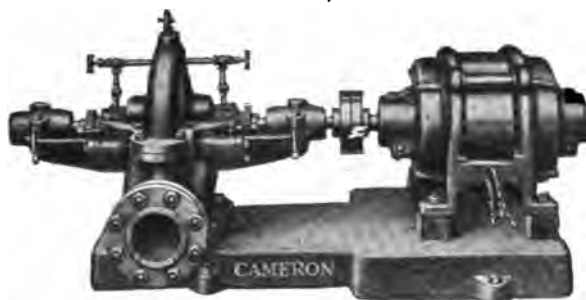
Maximum monthly or yearly output of a plant depends upon a high daily record. An apparatus such as the Weightometer makes this possible.

It is claimed that the machine in question automatically records the weight of material without complicating the conveyor system, and that apparatus in use for nearly two years has proved satisfactory.

The Weightometer consists of a pair of weighing levers, and a steel-yard or beam similar to the usual platform scale, but of special design so that a short section or portion of the conveyor can be suspended from the weighing levers.

The weight of the load on this suspended portion of the conveyor, regardless of its distribution, is at any instant automatically counter-balanced by the buoyancy of a cylindrical iron float suspended from near the long end of the weighing beam and partly immersed in a bath of mercury. Any increase or decrease of load on the levers will either raise or lower the float in the mercury until the loss or gain in buoyancy compensates for the variation in load.

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The function of this float is to insure that the movement of the beam from its zero position, or position when the conveyor is empty, be proportional to the weight of material at any instant on the suspended portion of the conveyor.

The extreme end of the beam is connected with a totalizing mechanical integrator, which derives its other factor from the travel of the conveyor by means of suitable gearing from a bend pulley on the return belt or a sprocket wheel if on a bucket conveyor.

This integrator continuously totalizes the product of two quantities, one proportional to the weight of material suspended and the other to the travel of this material. The result, therefore, represents the total weight of material and is plainly indicated by a register in units and tenths of units of either a short ton, long ton, or metric ton.

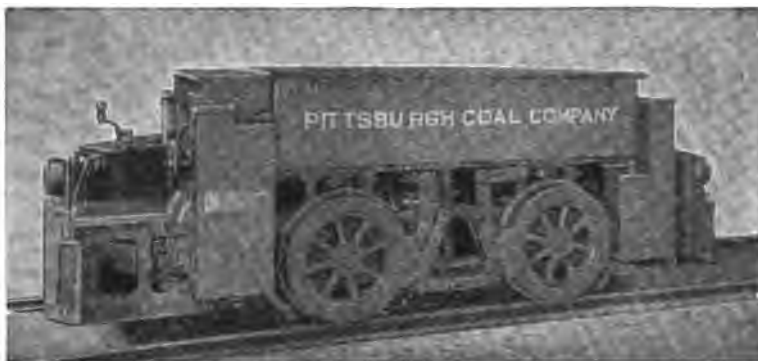
For cases where the material handled adheres to the conveyor in a varying amount, an attachment is added that automatically counterbalances the variable weight of the empty conveyor. This avoids frequent adjustment to meet the changes in the weight of the empty conveyor and material that may adhere thereon.

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JULY, 1915



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AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

Mr. _____
(Name in Full)

Occupation _____

Address _____

is hereby proposed by the undersigned, as a _____

of the American Institute of Mining Engineers.

} Signatures of three
Members or
Associates.

Place of birth _____ Year of birth _____

Education, general and technical, when, where and how acquired,
with degrees, if any.

Dates	

[illegible]

.....

Dated _____ *191*

ARTICLE II.—MEMBERS.

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

Harvard College Library
Aug. 14, 1916.
Bequest of
Erasmus Darwin Leavitt.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 103

JULY

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTSKY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STROUGHTON, Editor, F. O. FRENCH, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

SAN FRANCISCO MEETING

The 111th meeting of the Institute for the presentation and discussion of technical papers will be held in San Francisco, Cal., Sept. 16 to 18, 1915. A special train for members of this Institute and for the members of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the Society of Naval Architects and Marine Engineers, and the International Engineering Congress, will be run from New York City to San Francisco, leaving New York on Sept. 9.

ONLY ONE OFFICIAL TRAIN TO SAN FRANCISCO

We wish to notify our members that the official tour arranged by a joint committee of the Engineering Societies, leaving New York on Sept. 9, and arriving in San Francisco on Sept. 15, is the only official tour in connection with the International Engineering Congress or with the meeting of the Institute in San Francisco.

NOMINATIONS FOR OFFICERS

The co-operation of the members of the Institute is earnestly sought by the Committee on Nominations, recently appointed by the Board of Directors, in its work of formulating a ticket for officers and places falling vacant in the Board of Directors in February, 1916. The Committee must present the official ticket of nominations to the Board of Directors at its October meeting. Members are, therefore, urged to send promptly their nominations for officers and Directors in order that the Committee may have an opportunity to give them full consideration.

The officers to be elected at the annual meeting in February, 1916, are: One officer, known as Director and President. Two officers, known as Director and Vice-President. Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N. B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, William L. Saunders (not eligible for re-election).

Past President, Charles F. Rand.

Vice-Presidents, Thomas H. Leggett, District 0; Fred W. Denton, District 4.

Directors, John W. Finch, District 7; John H. Janeway, District 0; Edward P. Mathewson, District 5; Joseph W. Richards, District 2; George D. Barron, District 0.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, Crocker Bldg., San Francisco, Cal.

COMMITTEE ON NOMINATIONS: F. W. BRADLEY, *Chairman*.

LOUIS S. CATES,
R. C. GEMMELL,

JAMES F. KEMP,
FRANK A. ROSS,

FRANK M. SMITH,
ARTHUR THACHER.

BOARD OF DIRECTORS

Meeting of June 4, 1915.—The President announced the appointment of the following Committee on National Reserve Corps of Engineers: Dr. Henry S. Drinker, Chairman; Arthur S. Dwight, and Warren A. Wilbur; also, the appointment of Arthur F. L. Bell as Chairman of the Committee on Petroleum and Gas, and of James F. Kemp as Chairman of the Committee on Mining Geology.

All of these appointments were approved.

It was voted that the following minute be placed on record and a copy sent, by order of the Board of Directors, to the family of John Birkinbine:

"The Board of Directors of the American Institute of Mining Engineers hereby records its sense of sorrow and loss in the death of John Birkinbine, a former President, and for many years an active member of the Institute, and contributor to its *Transactions*.

"Mr. Birkinbine's eminence as an engineer, author and editor, intimately connected with the industrial progress of the United States, brought him into contact with the leaders of science and the captains of industry, among whom his genial and estimable personal character won for him many friends; and his departure in the prime of strength and usefulness is mourned by a wide circle."

It was voted that a joint session of this Institute with the American Electrochemical Society be held at the 111th (San Francisco) Meeting.

Upon the recommendation of the Committee on Membership, 29 Members, 2 Associates, and 16 Junior Members were elected.

The resignations of 4 members of the Institute were accepted with regret.

Three members were dropped for non-payment of dues.

At the dinner preceding the meeting of the Board of Directors on June 4, 1915, Huan Yi Liang and B. Atwood Robinson were the personal guests of the President of the Institute. Mr. Liang is President of the Government Lead Mine in Hunan, Chief Engineer of Wah Chang Antimony Mining & Smelting Co., Yu Tung Tin Mining Co., and Chi Dai Lead Mining Co. Mr. Robinson is Honorary Advisor to the Ministry of Agriculture and Commerce of China. These men are members of a commission visiting this country as representatives of the Chamber of Commerce of China.

President Saunders, acting as Toastmaster, called upon the following men, who responded by extending a welcome to the distinguished guests, and by paying tribute to the Republic of China, which they represent: Gano Dunn, President of the United Engineering Society, and Past President of the American Institute of Electrical Engineers; Walter Renton Ingalls, President of the Mining and Metallurgical Society of America; and Sidney J. Jennings, Vice-President of the Institute.

Mr. Liang in his reply pointed out that while China is extremely rich in natural resources, at present she is poor by reason of the fact that the development of these resources has been handicapped by lack of capital and of experienced engineers. China affords wonderful opportunities, he said, for American mining engineers and financiers.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period May 10 to June 10, 1915:

Robert M. Keeney, Somersville, Conn.
Emilio Mosonyi, San Salvador, C. A.
Junjiro Nagazumi, Fukuoka, Japan.
Francis R. Pyne, Elizabeth, N. J.
F. C. Alsdorf, Los Angeles, Cal.
W. J. Loring, London, England.
R. A. Bull, Granite City, Ill.

C. A. Moffett, Birmingham, Ala.
A. W. G. Wilson, Ottawa, Canada.
G. H. Glover, Jr., Short Hills, N. J.
R. W. Brock, Vancouver, B. C.
J. L. W. Birkinbine, Philadelphia, Pa.
Roy H. Allen, Jamesburg, N. J.

Francis Andrew Thomson received the degree of Master of Science from the Colorado School of Mines at the recent commencement exercises. The subject of his thesis was Stamp Milling and Cyaniding.

Benjamin Magnus, General Manager of the Mt. Morgan gold mine and Consulting Engineer to the Electrolytic Refining & Smelting Co., Ltd., has resigned and will return to America. He expects to arrive in New York City in September, and will make his headquarters care of H. M. Toch, 320 Fifth Avenue, New York.

Chester A. Orr has resigned his position as Superintendent of the Central Furnace of the American Steel & Iron Co., Cleveland, Ohio, to become Manager of the Cleveland Metal Products Co.

Edward W. Parker has resigned from the U. S. Geological Survey to accept an appointment as head of a bureau to be formed by the anthracite mining companies.

Ralph Hayden, formerly Superintendent of the slime plant of the Anaconda Copper Mining Co., has been appointed Superintendent of the regrinding and flotation plants at the Washoe Reduction Works, and will also have charge of the new slime plant upon its completion.

Robert Rhea Goodrich was granted the Degree of Doctor of Philosophy by Columbia University. Dr. Goodrich conducted an investigation and presented a dissertation on The Hydro-electrolytic Treatment of Copper Ores.

Horace V. Winchell has moved his office to 826 First National-Soo Bldg., Minneapolis, Minn.

Henry S. Munroe was the guest of honor at a dinner given by the Alumni of the Columbia School of Mines at the Chemists' Club in New York on the evening of May 28, 1915.

Rush J. White has retired from the staff of the Federal Mining & Smelting Co., and has opened an office as consulting engineer at Wallace, Idaho.

W. F. Ferrier, of Toronto, Ont., had the degree of Doctor of Science (honoris causa) conferred upon him by the University of Alberta at its convention on Apr. 28, 1915.

H. D. McCaskey has been appointed to succeed **Edward W. Parker** as Chief of the Division of Mineral Resources in the U. S. Geological Survey.

Walter B. Chidester is doing drafting work in connection with the remodeling of the concentrator of the Mountain Copper Co., Keswick, Cal.

William Wraith has been made General Manager of the International Smelting & Refining Co., at Tooele, Utah.

J. B. Tyrrell, of Toronto, was elected President of the Geological Section of the Royal Society of Canada at its annual meeting, held in Ottawa on May 25 to 27, 1915.

Robert M. Keeney has resigned his position as Superintendent of the mill of the Baker Mines Co., Cornucopia, Ore., to accept a position with the Magnesium Manufacturing Corporation, Rumford, Me.

Paul W. Gaebelein has been made Superintendent of the new mill of the Baker Mines Co., Cornucopia, Ore.

Albert E. Wiggin, formerly concentration engineer for the Anaconda Copper Mining Co., has been appointed Superintendent of Concentration, and will have charge of all concentration operations of the company in Montana.

Charles R. Wraith, formerly chemist in the laboratory at the Washoe Reduction Works of the Anaconda Copper Mining Co., has been placed in charge of the company's exhibit at the Panama-Pacific International Exposition.

F. L. Bosqui has opened an office at 1 London Wall Bldg., London, England, as a consulting metallurgical engineer.

Frederick G. Clapp of Pittsburgh has recently returned from a year and a half spent in geological explorations in China. His address is as before—331 Fourth Avenue, Pittsburgh, Pa.

AFFILIATED STUDENT SOCIETIES AND COLLEGE NOTES

Colorado School of Mines

The forty-first annual commencement exercises of the Colorado School of Mines were held at Golden, Colo., on Friday, May 28, 1915. Dr. Lucien I. Blake delivered an address on Epochs in Science. The degree of Engineer of Mines was conferred upon 30 men, and the degree of Master of Science on one man.

At the recent annual meeting of the Mining Engineering Society, Massachusetts Institute of Technology, officers were elected as follows: President, K. M. Sully; Vice-President and Treasurer, D. M. McRae; Secretary, H. Solakian.

The Student Auxiliary Society of the A. I. M. E. at the University of Kansas, Lawrence, Kan., has elected officers as follows for the year 1915-16: President, Harry E. Crum; Vice-President, Leland E. Fiske; Secretary and Treasurer, Ben A. Sweeney.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Position for a single man experienced in cyanide mill construction and operation at a small mine in Nicaragua. Salary \$150 per month and expenses. No. 39.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, desires position in ore dressing, either experimental or operating. Has had practical experience in flotation and other special methods. No. 228.

Member, technical graduate, 10 years' experience in Belgian coal mines, five years' experience in American coal mines in connection with reorganization of companies and opening of new coal fields. Aged 37. Speaks Spanish. No. 229.

Member, graduate engineer, aged 44, with over 20 years' experience in connection with large coal and coke operations in different parts of the country, desires position as general superintendent of large coal company, or will take an interest with the management in a smaller company. No. 230.

Member, technical graduate, with wide experience in mining and management, and as engineer in construction of plants, desires position as manager or consulting engineer for mining company, or responsible persons. No. 231.

Member, with many years' experience in management, purchasing, and accounting, would like position as office manager, secretary or treasurer of good company or firm. No. 232.

Junior member, Columbia graduate, young, ambitious; Arizona experience in surveying, assaying, cyaniding, sampling, and accounting; Spanish. Desires good opportunity for advancement. Southwest or Latin America preferred. No. 233.

Member, aged 30, unmarried, technical graduate in geology, with seven years' experience in the exploration and development of metalliferous ore bodies; who has been employed in Mexico, Michigan, New Mexico, South America, and Arizona; desires a position as geologist for a large mining or exploration company. Can go anywhere after Nov. 10, 1915. No. 234.

Recent graduate, single, has specialized in oil and gas course, and has experience in the oil fields. Will go anywhere as geologist for oil company or investor. References given. No. 235.

SOUTHERN CALIFORNIA LOCAL SECTION

*Executive Committee*THEODORE B. COMSTOCK, *Chairman*SEELEY W. MUDD, *Vice-Chairman*FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg.,
Los Angeles, Cal.

A. B. CARPENTER

C. COLCOCK JONES

A meeting of the Southern California Local Section, in honor of the presence of Secretary Bradley Stoughton, was held at luncheon in a private dining room of the Sierra Madre Club on Friday, May 21, 1915, 27 members and guests being present. In the absence, through illness, of the Chairman, Dr. Comstock, the vice-Chairman, Seeley W. Mudd, presided. After the luncheon, Secretary Stoughton made a short address on the activities of the Institute. George S. Rice, of the U. S. Bureau of Mines, who was present, then spoke on the activities of his Bureau. Local Secretary F. J. H. Merrill gave a description of the activities of the Southern California Section and its progress, and reported the latter as substantial and its condition as very good.

After some informal remarks by other members present the meeting adjourned.

F. J. H. MERRILL, *Secretary*.

COLUMBIA LOCAL SECTION

*Executive Committee*FRANK A. ROSS, *Chairman*RUSH J. WHITE, *Vice-Chairman*L. K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane,
Wash.

FREDERIC KEFFER

FRANCIS A. THOMSON

Holds four Sessions during year

Annual Meeting in September or October

A luncheon in honor of Bradley Stoughton, Secretary of the Institute, and Prof. Joseph W. Richards was tendered jointly by the Columbia Local Section, the Spokane Mining Men's Club, and the Spokane Stock Exchange at the Hotel Spokane, Spokane, Wash., on May 5, 1915. Frank A. Ross, Chairman of the Columbia Section, presided, and the following is an extract from his remarks:

"In gatherings of this kind it is scarcely necessary to dwell at length upon the history and work of our international association. Mining is a basic industry and the American Institute of Mining Engineers is the technological embodiment of its guiding genius. Founded 44 years ago by a small coterie of far-sighted engineers, the Institute has so thrived and grown that now its year book is glorified by hundreds of names made famous by wordly achievement. Scarcely a mining camp in the world but harbors one or more of its members, while its 50 fat volumes of *Transactions* are standard reference works of the profession. During 27 long years, that secretary of

secretaries, Dr. Rossiter W. Raymond, labored incessantly to make these *Transactions* models of their kind and it was owing mainly to his individual efforts, patience, and sagacity that our Institute arose from a struggling association to the dignity of a great institution that will endure as long as there is mining work to do.

But, in 1911, feeling the weight of advancing years, Dr. Raymond accepted the post of Secretary Emeritus to the Institute, reluctantly shifting this life-work of his to younger shoulders, which to-day are those of his worthy successor, our distinguished guest, who is now devoting his own life to an amplification of the work of his illustrious predecessor. Hence, on behalf of the Columbia Section of the American Institute of Mining Engineers, the Spokane Mining Men's Club, and the Spokane Stock Exchange, I take great pleasure in welcoming to Spokane and the Pacific Northwest Mr. Bradley Stoughton, our Secretary, and his associate, Dr. Joseph W. Richards, Professor of Metallurgy in Lehigh University and Secretary of the American Electrochemical Society."

L. K. ARMSTRONG, *Secretary*.

COMMITTEE ON CLASSIFICATION OF TECHNICAL LITERATURE

Delegates from about 20 national technical and scientific societies met in the Engineering Societies Building on May 21, 1915, to perfect a permanent organization, the purpose being to prepare a classification of the literature of applied science which might be generally accepted and adopted by these and other organizations. There was a generally expressed opinion that such a classification, if properly prepared, might well serve as a basis for the filing of clippings, for cards in a card index, and for printed indexes, and that the publishers of technical periodicals might be induced to print against each important article the symbol of the appropriate class in this system, so that by clipping these articles a file might be easily made which would combine in one system these clippings, together with trade catalogues, maps, drawings, blue prints, photographs, pamphlets, and letters classified by the same system.

By request, William P. Cutter, Librarian of the Engineering Societies Library, and a delegate from the American Institute of Mining Engineers, read a paper on The Classification of Applied Science.

Permanent organization was effected by the election of the following officers: Chairman, Fred R. Low; Secretary, William P. Cutter; Executive Committee, the above, with Edgar Marburg, H. W. Peck, and Samuel Sheldon.

The name adopted for this organization is, Joint Committee on Classification of Technical Literature.

MEMBERS NEEDING EQUIPMENT

It is with pleasure that we announce the receipt of a number of inquiries during the past month from those of our members needing assistance in locating manufacturers of various equipment. For each inquirer we have notified a number of firms best qualified to fill his needs. This one letter, instead of a dozen, secured the desired information, saved time and trouble, and proved the helpfulness of the A. I. M. E. to its members.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS**PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining and Metallurgy**

ELECTRIC MINE SIGNALLING INSTALLATIONS. By G. W. L. Paterson. London, 1914.

PRIMER ON EXPLOSIVES FOR METAL MINERS AND QUARRYMEN. Bull. 80, U. S. Bureau of Mines. Washington, 1915.

PERMISSIBLE EXPLOSION-PROOF ELECTRIC MOTORS FOR MINES; CONDITIONS AND REQUIREMENTS FOR TEST AND APPROVAL. Technical Paper 101, U. S. Bureau of Mines. Washington, 1915.

METALLURGICAL TREATMENT OF THE LOW-GRADE AND COMPLEX ORES OF UTAH. Technical Paper 90, U. S. Bureau of Mines. Washington, 1915.

PRELIMINARY CONCENTRATION TESTS ON CUYUNA ORES. Bull. 3, Minnesota School of Mines Experiment Station. Minneapolis, 1915.

Mining Law and Mine Valuation

NEW MINING LAWS OF ALASKA, 1913. By G. D. Emery.

GREAT BRITAIN. Home Office. Law relating to mines under the Coal Mines Act, 1911. London, 1914.

STUDY OF METHODS OF MINE VALUATION AND ASSESSMENT. Bull. 41; Wisconsin Geological and Natural History Survey. Madison, 1914.

Chemistry and Analysis

CHEMISTRY OF THE OIL INDUSTRIES. By J. E. Southcombe. London, 1913.

CHEMISTRY OF CYANOGEN COMPOUNDS. By H. E. Williams. London, 1915.

UNITED STATES STEEL CORPORATION. Methods for the Commercial Sampling and Analysis of Alloy Steels, 1915. (Gift of Carnegie Steel Co.)

Geology, Mineralogy, Mineral Production

ARIZONA. MINERAL DEPOSITS OF THE SANTA RITA AND PATAGONIA MOUNTAINS. Bull. 582, U. S. Geological Survey. Washington, 1915.

BRITISH COLUMBIA. GEOLOGY OF FRANKLIN MINING CAMP. Memoir 56, Canada Mines Department. Ottawa, 1915.

BRITISH COLUMBIA. MINERAL RESOURCES OF THE ATLIN MINING DIVISION. Bull. 2, British Columbia Bureau of Mines. Victoria, 1915. (Gift of British Columbia Bureau of Mines.)

BRITISH COLUMBIA. MINERAL RESOURCES OF THE SKEENA MINING DIVISION. Bull. 3, British Columbia Bureau of Mines. Victoria, 1915. (Gift of British Columbia Bureau of Mines.)

BRITISH COLUMBIA. MINERAL RESOURCES OF A PORTION OF THE Omineca Mining Division. Bull. 4, British Columbia Bureau of Mines. Victoria, 1915. (Gift of British Columbia Bureau of Mines.)

CALIFORNIA-NEVADA. MINING DISTRICTS (some) IN NORTHEASTERN CALIFORNIA AND NORTHWESTERN NEVADA. Bull. 594, U. S. Geological Survey. Washington, 1915.

CANADA. GEOLOGY OF THE NORTH AMERICAN CORDILLERA AT THE FORTY-NINTH PARALLEL, pt. 2, 3. Memoir 28, Canada Mines Department. Ottawa, 1912.

COLORADO. GEOLOGY AND COAL RESOURCES OF NORTH PARK. Bull. 596, U. S. Geological Survey. Washington, 1915.

NORWAY. STATISTISK AARBOK FOR KONGERIKET NORGE. 34 Aargang, 1914. Kristiania, 1915. (Gift of Norvège Bureau Central de Statistique.)

WEST VIRGINIA GEOLOGICAL SURVEY. County Reports, 1915. (Boone County, with maps.) Wheeling, 1915.

[NOTE.—This publication is described as a detailed report on Boone County, by C. E. Krebs, containing 648 pages + xviii of introductory matter, and illustrated with 43 half-tone plates and 3 figures or zinc etchings in the text; also a case of two maps covering the topography and geology of the entire area in one sheet. The soil map is attached to the accompanying Soil Report. In addition to the detailed description and revision of all the rich coal beds and other geologic formations exposed in these counties, the geologic map gives the structure contours and outcrops of the celebrated No. 2 Gas Coal, as also that of several other valuable coal beds, along with many new sections, analyses, etc. Price, with case of maps, delivery charges paid by the Survey, \$2. Extra copies of geologic map, \$1 each, and of the topographic map, 50 cents each. State Geologist, Morgantown, W. Va.]

Non-Metallic Minerals

CLAY AND SHALE DEPOSITS OF THE WESTERN PROVINCES. Memoir 55, Canada Mines Department. Ottawa, 1915.

NOTES ON CLAY DEPOSITS NEAR McMURRAY, ALBERTA. Bull. 10, Canada Mines Department. Ottawa, 1915.

COAL RESOURCES OF DISTRICT I (LONGWALL), ILLINOIS. Bull. 10, Illinois Coal Mining Investigations. Urbana, 1915.

COAL RESOURCES OF DISTRICT VII. (Coal No. 6 West of Duquoin anticline.) Bull. 11, Illinois Coal Mining Investigations. Urbana, 1915.

FELDSPAR AND MICA DEPOSITS OF GEORGIA, PRELIMINARY REPORT. Bull. 30, Georgia Geological Survey. Atlanta, 1915.

CONDENSATION OF GASOLINE FROM NATURAL GAS. Bull. 88, U. S. Bureau of Mines. Washington, 1915.

INVESTIGATION OF THE PEAT BOGS AND PEAT INDUSTRY OF CANADA, 1911-12. Bull. 9, Canada Mines Department. Ottawa, 1914.

POTASH IN THE TEXAS PERMIAN. Bull. 17, University of Texas. Austin, 1915. (Gift of the University of Texas.)

General

- CENTRIFUGAL PUMPS.** By R. L. Daugherty. New York, 1915.
FUEL, GASEOUS, LIQUID AND SOLID. By J. H. Costs and E. R. Andrews. London, 1914.
EMERY'S MINER'S MANUAL. Ed. 2. Chicago, 1912.

Company Reports

- WALLAROO & MOONTA MINING & SMELTING Co., LTD.** Annual Report, 25th, 1914. (Gift of Company.)
GEDULD PROPRIETARY MINES, LTD. Annual Report, 15th, 1914. (Gift of Hill Publishing Co.)
MAY CONSOLIDATED GOLD MINING Co., LTD. Annual Report, 1914. (Gift of H. P. C.)
MODDERFONTEIN DEEP LEVELS, LTD. Annual Report, 15th, 1914. (Gift of H. P. C.)
PRINCESS ESTATE & GOLD MINING Co., LTD. Annual Report, 1914. (Gift of H. P. C.)
TUDOR GOLD MINING Co., LTD. Annual Report, 15th, 1914. (Gift of H. P. C.)
VAN DYK PROPRIETARY MINES, LTD. Annual Report, 1914. (Gift of H. P. C.)

Trade Catalogues

- BUTCHART, W. A., DENVER, COLO.** Bull. 2, Butchart concentrating table. April, 1915.
FRANKLIN MOORE Co., WINSTED, CONN. Chain blocks, trolleys, hand-power cranes. 10 pp.
MOYLE, E. H., ENGINEERING & EQUIPMENT Co., LOS ANGELES, CAL. Moyle improved combination ore feeder and ore bin gate. 1915.
NATIONAL TRANSIT Co., OIL CITY, PA.
 Bull. 2A. Foam pumps.
 Bull. 5. Tools for pipe-line construction.
 Bull. 10. Straight line individual well pumping rig.
 Bull. 102. Steam pumps (Klein piston type).
 Bull. 103. Steam pumps (separate chest type).
 Bull. 404. Gas engines, horizontal tandem.
PERRIN, WM. R. & Co., CHICAGO, ILL. Description of filter presses manufactured by company. 4 pp.
SPARTA IRON WORKS Co., SPARTA, WIS. Sludge Bucket. May, 1915.
SUPPLEE-BIDDLE HARDWARE Co., PHILADELPHIA, PA. Monel-Metal. May, 1915.
WEBSTER MFG. Co., TIFFIN, OHIO. Webster Method. March, 1915.
TURBO-GEARS. Turbo-Gear Co., Indus Bldg., Baltimore, Md. Catalogue A introduces this system of gearing for high and low speed machines. General features of design, illustrations and table of horsepower transmitted by their gears are given.
CAMERON PUMPS. A. S. Cameron Steam Pump Works, 11 Broadway, New York. Bulletins Nos. 104, 150, 151, 152, 153 and 300, illustrate and describe their various styles of pumping apparatus. Detailed information is given on heads, capacities, speeds as well as other data of interest to pump users.
DERRICK EXCAVATORS. Union Iron Works, Newark St., Hoboken, N. J. 24 pp. Illustrated, and describes attachment for derricks, which enables the derrick to perform the same work as a steam shovel or dipper dredge. Views of apparatus in actual operation, table of sizes and other data are given.
HOISTING AND CABLEWAY EQUIPMENT. National Hoisting Engine Co., 2 Manor Ave., Harrison, N. J. Two catalogues, well illustrated, describe their line of hoists, cableways, etc., and give instances of installations. Specifications, and list of hoist and cableway accessories and repair parts are given.
PILE HAMMERS. National Hoisting Engine Co., 2 Manor Ave., Harrison, N. J. 20 pp. Well gotten up and illustrated. Describes their National steam pile hammer. Full information and various installations given.
MOORE SLIMES PROCESS. Moore Filter Co., 117 Broadway, New York, N. Y. 31 pp. Catalogue well illustrated and describes four methods of cyanidation by the Moore process.
ENGINEERING INSTRUMENTS. Hanna Mfg. Co., Troy, N. Y. 108 pp. Fully illustrates and describes products of this company. Prices, weights, etc., are given, and other information of interest to the user of scientific instruments.

- PUMPS.** Goulds Mfg. Co., Seneca Falls, N. Y. Complete set of bulletins, illustrating and describing their many styles and assortments. Complete specifications and information of interest to pump users are given.
- MONEL METAL RODS.** Supplee-Biddle Hardware Co., 501 Commerce St., Philadelphia, Pa. 8 pp. Bulletin No. 5 describes new process of producing semi-cold rolled rods. Prices and stock of rods and sheets are given.
- PAINTS.** Toch Brothers, 324 Fifth Avenue, New York, N. Y. Complete set of catalogues and literature illustrating and describing their products for weather, rust, acid and water proofing materials needing such protection. Interesting data are given for the user of paint.
- HARDINGE CONICAL MILL.** Hardinge Conical Mill Co., 50 Church St., New York, N. Y. Address as above instead of 52 Broadway given in the June *Bulletin*.
- "CASCADE" PUMP.** Van Noughys Machine Works, 49 Liberty St., Albany, N. Y. 2 pp. Illustrated folder of their portable diaphragm pumper for use in trench and other work requiring a portable machine. Specifications and other data given of interest to pump users.
- WAGON LOADER.** George Haiss Mfg. Co., Inc., Rider Ave., New York, N. Y. 8 pp. Describes and illustrates their machine for use in handling coal, sand, gravel, broken stone, etc. Gives cost data to prove economy of their loader, together with detailed comparison between hand and machine loading.
- LAWSON LOOP-LINE TRAMWAYS.** Ambursen Co., 63 Broadway, New York, N. Y. 42 pp. Bulletins A and B illustrate and describe their apparatus for moving any kind or quantity of tonnage from a few hundred feet to any number of miles, at an agreed price per ton. Detailed illustrations show method and means of transporting and dumping material. Bulletin B discusses the right and wrong way of cable transportation.
- MESSITER CONVEYOR SCALES.** Electric Weighing Co., 184 13th Ave., New York, N. Y. 18 pp. Describes and illustrates their apparatus for weighing material while being carried on belt, pan and bucket conveyors, with recorder in manager's office.
- "CASE" METALLURGICAL FURNACES.** Denver Fire Clay Co., Denver, Colo. 80 pp. Illustrates and describes in detail up-to-date furnace equipments as a result of 40 years' experience in their construction. Several instances of economical operation are given. Cost data, partial list of users and other data of interest are given.

Notes on Trade Literature

The Ingersoll-Rand Co., 11 Broadway, New York, N. Y., have recently issued two new catalogues, forms 3031 and 4034. The former is descriptive of the new Ingersoll-Rogler Class "FR-1" steam-driven single-stage straight-line air compressors. The catalogue is profusely illustrated, showing the details of the machine in section. Among the principal features of construction may be mentioned the Ingersoll-Rogler air valves in the air cylinder and the balanced piston steam valve, with automatic cut-off control, in the steam cylinder. It is claimed that the air valves are silent in operation and long lived, operate entirely independent of any driving mechanism, and require practically no attention, and that the steam valve represents the most approved design, permitting the use of the highest steam pressures and superheated steam.

Form 4034 covers the type No. 26 Leyner-Ingersoll water drill. This drill resembles the larger successful Leyner-Ingersoll drill, No. 18, but, it is, however, considerably lighter in weight and intended for smaller diameter and shorter depth holes.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period May 10 to June 10, 1915:

Members

- ANDREEN, HARRY MAYO, Millman and Assayer..... Thane, Alaska.
 BANKS, HUBERT RAYMOND, Min. Engr., Mill Staff,
 Consolidated Arizona Smelting Co., Humboldt, Ariz.
 BASSETT, ARTHUR F., Min. Engr., Care Consolidated Arizona Smelting Co.,
 Humboldt, Ariz.
 BLACKNER, LESTER A., Stope Engr.... Care Ray Consolidated Copper Co., Ray, Ariz.
 BUSHNELL, C. E., Draftsman..... Butte & Superior Copper Co., Butte, Mont.
 CAVAGNARO, DAVID AMILE, Min. Engr., Genl. Supt., Wisconsin North Fork
 Gravel Mines, Forest, Sierra Co., Cal.
 COGHILL, WILLIAM HAWES, Met., Ore Testing Laboratory, 3705 Hueco St.,
 El Paso, Tex.
 CORBET, EDWARD B., Min. Engr., 1201 First National Bank Bldg., San Francisco, Cal.
 FERRIS, ALBERT LARUE, Mine Surveyor, Metcalf Division, Arizona Copper Co.,
 Metcalf, Ariz.
 GOULD, CHARLES NEWTON, Geol. Engr., 1218 Colcord Bldg., Oklahoma City, Okla.
 HALL, MORTIMER LOUIS, Min. Engr., Engr., Beatson Copper Co., Latouche, Alaska.
 HEINDL, ALEXANDER J., Min. Engr., Bodaibo, Province of Irkutsk, Siberia, Russia.
 HOGAN, JOHN E., Mine Foreman..... North Star Mines Co., Grass Valley, Cal.
 KELLY, FRANK JOSEPH, Min. Engr..... Mispah, Alberta, Canada.
 LANGENBERG, FREDERICK C., Asst. in Metallurgy, Harvard Univ., Rotch Bldg.,
 Cambridge, Mass.
 LARSH, WALTER STUART, Min. Engr., Chile Exploration Co., Chuquicamata, Chile.
 LARSON, CLARENCE L..... 320 So. Fifth East, Salt Lake City, Utah.
 McNABB, JAMES C., Assayer and Chem., Care Consolidated Arizona Smelting Co.,
 Humboldt, Ariz.
 MACKEY, ROBERT W., Min. Engr..... Smuggler, Colo.
 MATHEWS, JOHN A., Genl. Mgr. Halcomb Steel Co., Syracuse, N. Y.
 MEAD, HARRY LEVI, Min. Engr..... 122 E. 36th St., New York, N. Y.
 MOORE, ELWOOD S., Geol..... 6 Rue Marguerin, Paris, France.
 MORGAN, WILLIAM LLEWELYN, Instructor, Illinois Miners' and
 Mechanics' Institute, East St. Louis, Ill.
 OLIVEIRA, EUZEBIO PAULO DE, Geol., Brazilian Geological Survey, Rio de Janeiro,
 Brazil.
 PAGE, JAMES MADISON, Mine Supt..... Sunnyside Coal Mining Co., Strong, Colo.
 PRATT, MORTON EDGAR, Min. Engr.; Supt., South American Development Co.,
 Guayaquil, Ecuador.
 PRIESTLEY, WILLIAM J., Supt., Armor Plate Dept., Bethlehem Steel Co.,
 So. Bethlehem, Pa.
 SCHIRMER, REBS EDWARD, Mine Mgr., Argo Mining, Drainage, Transportation &
 Tunnel Co., Idaho Springs, Colo.
 SEAGRAVE, WILLIAM HENRY, Mine Mgr.... Kennecott Mines Co., Kennecott, Alaska.
 TAYLOR, CHARLES H., Geol.; Prof. of Geology, University of Oklahoma, Norman, Okla.

Associate Member

- FARGO, LIVINGSTON WELLS..... University Club, Chicago, Ill.

Junior Members

- HUNT, JOHN EDWARD, Student..... 245 E. Granite St., Butte, Mont.
 NICOLSON, CLYDE WALLACE, Student..... 233 College Ave., Houghton, Mich.
 RUEBEL, ERNST HERTEL, JR., Student..... Missouri School of Mines, Rolla, Mo.

Change of Status—Associate Member to Member

- CAMPHUIS, GEORGE A., Min. Engr..... Room 614 Mills Bldg., El Paso, Tex.

Junior Member to Member

- LEACH, ROY, Cyaniding..... Melones, Calaveras Co., Cal.

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period May 10 to June 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Jose Nicandro Acosta, Gleeson, Ariz.

Proposed by E. Grebe, John D. Wanvig, Jr., John E. Penberthy.

Born 1876, Fresnillo, Zac., Mexico. Elementary studies, Mapimi. Advanced Studies. 1892-95, Instituto Cientifico y Literario, Chihuahua. 1895-98, Professional, Colegio de Minería, Mexico City. 1898-99, Compania Minera "El Carrizo," Chihuahua. 1899-1900, Negociacion Minera "Cree y Varela," Chihuahua. 1900-03, F. Stallforth y Hno. Sucre and Compania, Parral. 1903-11, Independent engineer and chemist. 1911-12, State Engr., Sonora. 1912-14, Assayer and Chem., Copper Queen Cons. Min. Co., Bisbee, Ariz. 1914-15, Assayer and Chem., Phelps, Dodge & Co., Tombstone, Ariz.

Present position: Min. Engr. and Chem., Shannon Copper Co.

C. Harry Benedict, Lake Linden, Mich.

Proposed by Allan J. Clark, Thomas T. Read, James MacNaughton.

Born 1876, Pittsburgh, Pa. 1897, Cornell Univ.; B. S.

Present position: 1898-1915, Met., Calumet & Hecla Min. Co.

James Malcolm Bonsall, Douglas, Ariz.

Proposed by P. P. Butler, Lawrence Addicks, Harold O. Hammond.

Born 1878, Morristown, N. J. 1902, Princeton; B. S. 1902-03, Sawyer-Mann Electric Co. 1904-05, Copper Queen Cons. Min. Co. 1911-12, Raritan Copper Refinery.

Present position: Chem., Copper Queen Reduction Works.

Ross Bowles, East St. Louis, Ill.

Proposed by H. A. Wheeler, Philip N. Moore, O. M. Bilharz.

Born 1871, Bedford, Pa. Employed since nine years of age. Education obtained by practice and study. 1891-1902, Cashier and purchasing, St. Joseph Lead Co., Bonne Terre, Mo. 1902-05, Fruin Bambrick Construction Co., St. Louis, railroad construction and street paving. 1906-15, Supt. of zinc smelter, Granby Min. & Smelt. Co., Neodesha, Kan.

Present position: Supt. of Smelters, Granby Min. & Smelt. Co. in charge of both Neodesha, Kan., and East St. Louis, Ill. Smelters.

Alexander M. Boyle, Guayquil, Ecuador.

Proposed by T. W. Mather, Roy C. Philpott, Rush M. Hess.

Born 1880, Gold Hill, Nev. General education acquired by hand. 1902-03, Math. and mechanics, Pa. 1903-07, Min. Engrg., Nev. General mining, milling, and

shop experience covering a period of several years before entering college. 1907-10, Miner, mechanic, shift boss, foreman and assistant superintendent with various stock mining companies. 1911, Asst. to Mgr., Trent Engineering Co., Reno, Nev. 1912, Master Mechanic, White Caps Leasing Co., Manhattan, Nev. 1913, Foreman, New Golden Crown Min. Co., Tonopah, Nev.

Present position: Master Mechanic, South American Development Co.

Bert S. Butler, Washington, D. C.

Proposed by Waldemar Lindgren, H. D. McCaskey, F. C. Schrader.

Born 1877, Gainesville, N. Y. 1905, Cornell Univ.; A. B.; 1907, A. M. 1904-05, Asst. in Geology. 1905-07, Instructor in Geology and Physical Geography, Cornell Univ. 1907-12, Asst. Geol., U. S. Geological Survey.

Present position: Geol., U. S. Geological Survey.

H. A. Clark, Douglas, Ariz.

Proposed by J. Owen Ambler, John C. Greenway, L. D. Ricketts.

Born 1871, Minneapolis, Minn. General education through graded schools; one year in high school, Minneapolis, Minn. Previous to 1900, journeyman machinist. 1900-05, Machinist and stationary engineer, Dell's Paper Co., Eau Claire, Wis. 1905-07, Master Mechanic, Itaska Paper Co., Grand Rapids, Minn. 1907-11, Machine-shop foreman, Assistant Master Mechanic, Master Mechanic, Oliver Iron Co., Canastota District, Minn.

Present position: 1911 to date, Master Mechanic, Construction Supt, and Smelter Supt., Calumet & Arizona Min. Co.

George Hamilton Cunningham, Anaconda, Mont.

Proposed by Albert E. Wiggins, Frederick Laist, E. P. Mathewson.

Born 1883, Kelly's Ford, Va. 1899-1901, Eastern View Academy, Culpeper, Va. 1906, Va. Polytechnic Inst. Blacksburg, Va.; B. S. in Mech. Engrg. 1907, M. E. 1908, Cornell University, Ithaca, N. Y.; M. E. A member of A. S. M. E. 1906-07, Instructor in Graphics, Va. Polytechnic Inst. 1908-09, Rodman and instrument man on construction, Tenn. Coal, Iron & R. R. Co., Ensley, Ala; and blast-furnace and steel-mill work. 1909, Draftsman, Niles Tool Works, Hamilton, Ohio. 1910-11, Draftsman, Va. Bridge & Iron Works, Roanoke, Va. 1911, Asst. Supt., Power & Mechanical Dept., Consolidation Coal Co., Jenkins, Ky. 1912-13, Asst. Prof. Mech. Engrg., Univ. of Montana, Missoula, Mont.

Present position: Drafting and smelter construction, Anaconda Copper Min. Co.

George M. Davies, Lansford, Pa.

Proposed by Edwin Ludlow, W. G. Whildin, W. B. Richards.

Born 1848, South Wales. Attended public school until nine years old; left to work in coal breaker; began to work in coal mines at 12 years of age and attended night school until 23 years of age. 1857-60, Slate picker in coal breaker. 1860-71, Doorboy, fan boy, driver, laborer, and loader in mines. 1871-78, Employed on engineer corps by J. S. McNair and John Righter, Hazleton, and continuation of work in mines. 1878-82, Contract miner in mines near Hazleton. 1882-1906, Contractor operating mine under lease from Lehigh Coal & Navigation Co., Lansford, Pa.

Present position: Contractor, steam shovel stripping coal, Lehigh Coal & Navigation Co.

C. O. Adolf Dittmann, Santiago de Chile, Chile.

Proposed by Berth Koerting, Mark R. Lamb, Thomas H. Barclay.

Born 1882, Magdeburg, Germany. 1888-1900, Realgymnasium, Magdeburg. 1900-04, Mining academy, Clausthal, Prussia; Diplom as Min. Engr. 1905-08, Univ. of Heidelberg, promoted as Doctor of Natural Science (Geology, Mineralogy and Chemistry). 1909-13, Mgr., Zinnstockwerk Geyersberg, Geyer, Saxony. 1912-13, Cons. Engr., Ehrenfriedersdorf Vereinigt Feld tin mine, Ehrenfriedersdorf, Saxony.

Present position: Cons. Engr., Beer, Sondheimer & Co., South American Branch.

George M. Douthett, Pittsburgh, Pa.

Proposed by Fred Crabtree, C. T. Criswold, L. L. Beeken.

Born 1894, Pittsburgh, Pa. 1914, Carnegie Institute of Tech.; B. S. 1915, Chemical analyst, Carnegie Steel Co., Duquesne, Pa.

Present position: Met. Engr., Pittsburgh Screw & Bolt Co.

Desiderius Ernyei, Budapest, Hungary.

Proposed by E. W. Moore, W. L. Saunders, George A. Howells.

Born 1881, Budapest, Hungary. 1903, Technical University in Budapest and Zurich; graduated mechanical engineer in Budapest. 1910, six months' practical mining field studies, private studies on mining engineering. 1904, General Electric Co., Lynn, Mass. 1907, Ganz & Co. Electric Works, Budapest, Dept. for supplying the machinery for mining purposes.

Present position: 1909 to date, Mgr., Ingersoll-Rand Co.

Simon Guggenheim, New York, N. Y.

Proposed by A. Eilers, H. A. Prosser, Willard S. Morse.

Born 1867, Philadelphia, Pa. Philadelphia public and high schools. Germany, France, Spain for two years, finishing education. Then in smelting business at Pueblo, Colo. 1888 at Philadelphia Smelter and continuously since then in Philadelphia Smelter and American Smelt. & Refin. Co. 1907-13, U. S. Senator from Colorado.

Present position: Chairman, Board of Directors, American Smelt. & Refin. Co.

Archibald R. Henley, Lansford, Pa.

Proposed by Edwin Ludlow, J. B. Warriner, W. G. Whildin.

Born 1886, England. To 1901, Public schools, West Pittston, Pa. 1907, Grad., Harry Hillman Academy, Wilkes-Barre, Pa. 1908-10, Special in Min., Lehigh University, So. Bethlehem, Pa. 1901-04; Draftsman, Exeter Machine Works, West Pittston, Pa. 1904-07 and 1910-11, Draftsman, Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1911-12, Draftsman, Lehigh Coal & Navigation Co., Lansford, Pa.

Present position: 1912 to date, Chief Mechanical Draftsman, Lehigh Coal & Navigation Co.

Charles R. Holzworth, Pittsburgh, Pa.

Proposed by Fred Crabtree, J. O. Willard, S. W. Gittins.

Born 1886, Youngstown, Ohio. 1910-14, Carnegie Institute of Tech.; B. S. 1908-10, Preparation at Y. M. C. A., Youngstown. 1902-04, Chem. lab., Republic Iron & Steel Co., Youngstown. 1904-06, Chem. lab. and furnace, Ohio Iron & Steel Co. 1906-08, Blast furnace, Republic Iron & Steel Co. 1908-14, Blast furnaces, Ohio Works, Carnegie Steel Co., Youngstown. 1914, Blast furnaces, Jones & Laughlin Steel Co., Pittsburgh, Pa.

Present position: Asst. to Supt., Eliza Furnaces, Jones & Laughlin Steel Co.

Samuel Leslie Hoyt, Minneapolis, Minn.

Proposed by W. H. Emmons, Elting H. Comstock, Peter Christianson.

Born 1888, Minneapolis, Minn. 1905, Central High School, Minneapolis, Minn. 1909, Univ. of Minnesota; E. M. 1909-11, Graduate work, Columbia. 1911-13, Graduate work and research, Technische Hochschule, Berlin, Germany. 1914, Ph. D. (Columbia) (granted 1915).

Present position: 1913 to date, Asst. Prof. Metallurgy, Univ. of Minnesota.

William Frederick Jahn, San Juancito, Honduras.

Proposed by John V. N. Dorr, P. M. McHugh, H. N. Spicer.

Born 1888, Germany. 1906, Grad., Winona High School, Winona, Minn. 1911, Grad., Univ. of Minn. School of Mines; E. M. 1911-12, Millman, Engr., Met., Mogul Min. Co., Pluma, S. D. 1912, Millman, Nevada Hills Min. Co., Fairview, Nev. 1913, Shift boss, mill and cyanide plant, of New York & Honduras Rosario Min. Co., San Juancito, Honduras.

Present position: Mill Supt., New York & Honduras Rosario Min. Co.

Clyde Homer Jay, Salida, Colo.

Proposed by R. D. George, Charles A. Chase, Fred Johnston.

Born 1877, Burton, Kan. 1897, Grad. State Preparatory School, Boulder, Colo. 1901, Grad., College of Engineering, Univ. of Colorado; B. S. Spent Junior college year at Colorado School of Mines. 1901-03, Chem., Colorado Fuel & Iron Co., Pueblo, Colo. 1903-04, Mill Foreman, later Chief Chem., General Metals Co., milling plant, Colorado Springs, Colo. 1904-05, Chem., Colorado Fuel & Iron Co., Pueblo, Colo. 1905-06, Chem., A. B. Smelter, Leadville, Colo. 1906, United States Smelt., Refin., & Min. Co., Bingham Canyon, Utah. 1906, Supt., Golden Sunlight & Ohio Min. Prop., White Hall, Mont. 1907, private engineering office, Rhyolite, Nev. 1908, Supt., Jennie Gold Min. Co., Gold Springs, Utah. 1909, Experimenting and perfecting my process for rapid cyanide treatment of slimes, for which I obtained U. S. Letters Patent in 1910.

Present position: Vice-Pres. and Mine Mgr., Giant Eclipse Cons. Mines Co.; Cons. Engr., New Discovery Min. & Mill. Co.; City Engineer, Salida, Colo.

Godfrey Samuel Kope, Johannesburg, Transvaal.

Proposed by William McC. Cameron, W. L. Honnold, C. E. Knecht.

Born 1887, Sydney, N. S. W., Australia. 1903-07, Cape Univ.; Grad., Mining Engineering. 1908, Obtained Mine Surveyor's Certificate. 1914, Obtained Mine Manager's Certificate. 1907, Machine stoping on New Gooch G. M. for one month during miners' strike. 1908, Sampling, Simmer & Jack East G. M. 1908, Sampling, Block & French Rand G. M. 1908-10, Sampling, shift bossing and surveying, Geduld Proprietary Mines. 1910-11, Developing, Simmer & Jack East G. M. 1911-12, Surveying and shift bossing, Geduld Prop. Mines. 1912, Shift bossing during shaft sinking, Springs Mines. 1912-13, Shift bossing, Geduld Prop. Mines. 1913-14, Surveying and clerk of works, construction of Reduction Plant, Modderfontein Deep Levels.

Present position: Surveyor, May Cons. G. M. Co.

Bernard Harold Lasky, San Francisco, Cal.

Proposed by D. M. Folsom, C. F. Tolman, Jr., James C. Ray.

Born 1892, San Francisco, Cal. 1913, Stanford Univ., geology and mining; A. B. 1912, underground and mill work, Pittsburgh Silver Peak, Blair, Nev. 1913, Engr., Eagle-Shawmut Min. Co., Shawmut, Cal. 1914, Engr., Presidio Min. Co., Shafter, Tex. 1915, Min. Engr.

Present position: Examination work.

Paul E. Leiss, Youngstown, Ohio.

Proposed by Karl Nibecker, Strickland Kneass, Jr., G. A. Reinhardt.

Born 1888, Robeson, Pa. 1903-07, Albright College, Myerstown, Pa. 1907-09, Penn. State College, State College, Pa.; B. S. 1909-10, Chem., Carnegie Steel Co., Braddock, Pa. 1910-11, Chem., Lake Superior Iron & Chemical Co., Detroit, Mich. 1911-12, Chem. Engr., Southern Railway Co., Alexandria, Va.

Present position: Testing Engr., Steam Eng. Dept., Youngstown Sheet & Tube Co.

Loring C. Lennox, Colorado Springs, Colo.

Proposed by Fred H. Bostwick, Franklin Moeller, Charles A. Chase.

Born 1881, Colorado Springs, Colo. 1901, Grad., Colorado Springs High School, 1906, Grad., Colorado College; S. B. 1906, In charge of work, Spring River Metals Co., Quapaw, Okla. 1907, Mgr., Country Boy Gold Min. Co., Breckenridge, Colo. 1908, Supt., Strong Gold Min. Co., Victor, Colo.

Present position: Mgr., Strong Min. Co.

Huan Yi Liang, Changsha, China.

Proposed by Walter Renton Ingalls, Benjamin B. Thayer, Hennen Jennings.

Born 1876, Changsha, China. 1906-08, Royal School of Mines, London. 1909-13, Ex-Vice-Pres., Yunnan Tin Co.; Pres., Mining School, Yunnan; Chief of the Yunnan government assay office; Pres. of Yunnan Antimony Co., of Canton Antimony Co.

Present position: Pres. of government lead mine in Hunan and Chief Engineer of Wah Chang Antimony Min. & Smelt. and Yu Tung Tin Min. Co., Chi Dai Lead Min. Co.

Charles McKnight, Jr., Sewickley, Pa.

Proposed by Karl Nibecker, G. A. Reinhardt, Stephen L. Goodale.

Born 1891, Sewickley, Pa. 1909, Lawrenceville School, Lawrenceville, N. J. 1916, Univ. of Pittsburgh, Pittsburgh, Pa.; B. S. 1910, Asst. Motor Inspector, Asst. Chem., Asst. Yard Boss, etc., Midland Steel Co., Midland, Pa. 1912, Drafting Room, Carrie Furnace, Carnegie Steel Co. 1913, Steam Engineering Dept., Youngstown Sheet & Tube Co.

Present position: Student, Univ. of Pittsburgh.

Albert Mendelsohn, Baltic, Mich.

Proposed by W. L. Saunders, George A. Howells, R. S. Carter.

Born 1889, Chicago, Ill. 1907-11, Columbia School of Mines; E. M. 1911-12, Efficiency Engr., Superior Copper Co. 1912, Copper Range Consolidated Co., Houghton Co., Mich., operating Baltic, Champion and Trimountain Mines.

Present position: Supt., Baltic Mine.

LeRoy Bradley Mitchell, Bisbee, Ariz.

Proposed by Roger T. Pelton, Alexander V. Dye, Gerald F. G. Sherman, Arthur Notman.

Born 1888, Bellevue, Ohio. 1911, Yale Univ.; Ph. B.

Present position: 1911 to date, Engr., Copper Queen Cons. Min. Co.

John Stanley Olmstead, Ajo, Ariz.

Proposed by L. D. Ricketts, F. L. Antisell, A. G. MacGregor.

Born 1886, Clifton Springs, N. Y. 1892-1900, Webster Grammar School, St. Paul, Minn. 1900-04, Central High School, St. Paul, Minn. 1904-08, School of Mines, Univ. of Minnesota, Minneapolis, Minn.; E. M. 1908-09, Eng. Dept., Calumet & Arizona Min. Co., Warren, Ariz. 1909-14, Geol. Dept., Calumet & Arizona Min. Co. 1912, Engr., Courtland Explorations, Courtland, Ariz., for Calumet & Arizona Min. Co.

Present position: 1914 to date, Asst. Engr. on experimental leaching, New Cornelia Copper Co.

Oliver C. Ralston, Salt Lake City, Utah.

Proposed by W. G. Woolf, D. A. Lyon, Robert S. Lewis.

Born 1886, Colorado Springs, Colo. 1910, Colorado College; B. S., E. M. 1904, Telluride Mill. Co. 1906, Portland Gold Min. Co. 1907, Cyanide work, Golden Cycle Min. Co. 1911-12, Assayer, M. G. Alexander, Leadville, Colo. Engr., McNair & Heitz, Leadville, Colo. Teaching chemistry, Leadville High School.

Present position: Asst. Met., U. S. Bureau of Mines.

George Egbert Rockefeller, Fairfax, Wash.

Proposed by George Watkin Evans, Joseph Daniels, Percy E. Wright, Harry Boyle.

Born 1885, Oskaloosa, Iowa. 1906, Electrician, Pacific Coast Coal Co. 1909, Mine Supt., Carbon Coal & Clay Co. 1912, Mine Supt., Tacoma Smelt. Co.

Present position: 1914 to date, Genl. Supt., Fairfax Mine, Inc.

C. E. Sampson, Shale, Cal.

Proposed by Walter Stalder, A. K. P. Harmon, Jr., B. K. Stroud.

Born 1883, Oakland, Cal. 1909, Stanford Univ., Cal., College of Electrical Engineering. 1910-11, North American Oil Co., Taft, Cal. 1911-14, Universal Oil Co., Wasca, Cal.

Present position: Supt., Recovery Oil Co.

John Vandemoer, El Paso, Tex.

Proposed by David L. C. Hover, G. S. Newell, R. B. Kepner.

Born 1877, Java, Dutch Indies. 1884, Public School, Nymegen, Holland. 1890, Private school, Zutphen, Holland. 1897, Grad., School of Tech., Amsterdam; M. E. 1897, Drawing room, Spangler Iron Works, Gronau, Germany. 1898, Sepp Iron Works, Enschede, Holland, laying out valve diagrams for comp. steam engines. 1899, Leyner Engineering Works, Denver, Colo. 1900, Denver Engineering Works, Denver, Colo. 1901, U. S. Gold Extraction Co., Florence, Colo. 1902, Hug Water Wheel Co., Denver, Colo. 1903, Zinc Smelter, Pueblo, Colo.; 1904, Leadville Smelter, Colo. 1906, Chihuahua Smelter, Mexico, A. S. & R. Co. 1908, In charge of all construction, Mazapi Copper Co., Saltillo, Mexico. 1910, Matehuala Smelter, A. S. & R. Co., San Luis Potosi, Mex. 1911, El Paso Smelter, El Paso, Tex. 1913, Chihuahua Smelter, charge of construction two lead blast furnaces, H. & H. roasting plant and Highline R. R. connection. 1915, Asarco Smelter, charge constructing crib dam.

Present position: Engr. and draftsman, American Smelt. & Refin. Co.

Herbert Michael Wilson, Sewickley, Pa.

Proposed by George S. Rice, Van H. Manning, W. A. Williams.

Born 1860, Glasgow, Scotland. 1875, Grad., Plainfield, N. J., High School. 1877, Cooper Inst., New York. 1881, School of Mines, Columbia Univ.; C. E. 1881-82, R. R. surveying, Mexico. 1883-87, Topographer, U. S. Geological Survey. 1888-92, Irrigation engineering. 1893-1906, Geographer, U. S. Geological Survey. 1907-15, Engr., U. S. Bureau of Mines.

Present position: Director, Coal Mine Insurance Assoc.

Herbert Betts Wolcott, Carthage, Mo.

Proposed by D. M. Folsom, C. F. Tolman, Jr., James C. Ray.

Born 1891, Carthage, Mo. 1915, Stanford University; A. B. Work in connection with my father, H. A. Wolcott, in connection with the Pittsburgh Missouri Lead & Zinc Co.; Pittsburgh Joplin Lead & Zinc Co.; Little Martha Min. Co.; Quaker Min. Co.

Present position: Student, Stanford University.

Associate Members

Henry Bornhoft, El Paso, Tex.

Proposed by Theodore Sternfeld, B. Hochschild, Otto Sussmann.

Born 1877, Hamburg, Germany. 1893, Left high class of Hamburg Seminary School. No technical schooling. 1896, Finished business training in Hamburg and subsequently held positions in commercial houses. 1899-1902, Negociacion Minera de Candelaria y Anexas, Pinos, Zac., Mexico. 1902-12, Cia. Minera de Penoles, Mapimi, in various capacities.

Present position: Asst. Gen. Mgr., Penoles Min. Co.

Cary R. Miller, Stockholm, Sweden.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1862, Millersburg, Ind. 1881, South Bend, Ind. 1887-93, Grand Trunk R. R. 1893-99, Lake Shore & Michigan Southern R. R. 1899-1900, Ryder & Conley. 1900-03, Fahr & Treff.

Present position: 1903 to date, Ingersoll-Rand Co.

Robert Rae, Douglas, Ariz.

Proposed by C. Legrand, Cleveland E. Dodge, Hylton H. Colley.

Born 1871, Glasgow, Scotland. Public schools of Glasgow, Scotland. 1898, Clerk, John Kirkwood & Sons, Glasgow, Scotland. 1900, Public accountant, Cuthbert, Menzies & Co., New York.

Present position: General Auditor, Phelps, Dodge & Co., Inc.

William Lingle Scantlin, Lead, S. D.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1884, La Fayette, Ind. 1907, Grad., Purdue Univ., School of Mech. Engrg.; B. S. 1907-08, Salesman, Railway Materials Co.

Present position: Sales Engr., Ingersoll-Rand Co.

Charles R. Watson, Cusiuhiriachic, Mexico.

Proposed by H. L. Hollis, D. P. Hynes, W. J. Wallace.

Born 1873, Hartford, Kan. 1890, Southern Kansas Academy, Eureka, Kan. 1890-93, Machinist apprentice, Santa Fe Ry., Nickerson, Kan. 1893-98, Journeyman machinist. 1898-1900, Shop foreman and erection, Grand Central Min. Co.; Creaton Colorado Min. Co., and Butters Syndicate Mines. 1900-02, Mine Supt., La Dura, Son., Mexico. 1902-05, Concentrator erection at La Bufa. Drilling "Dios Padre" mine, Trinidad, Son. Mgr., Cia. Minera La Gloria, La Bufa, Son. 1905-08, Examinations, M. M. New Sabinas Co., La Sabinas, Coahuila. 1909-10, Mgr., Edna Mines Co.

Present position: Mgr., Cusi Min. Co.

Junior Members

John Lane Applegate, Youngstown, Ohio.

Proposed by Charles H. Fulton, J. Burns Read, Zay Jeffries.

Born 1889, Youngstown, Ohio. 1913, With field engineer, Fred H. Hubbard, Carnegie Steel Co., Youngstown.

Present position: Student, Case School of Applied Science.

Ambrose Hammet Burroughs, Jr., Irvington, N. Y.

Proposed by Robert Peele, Arthur L. Walker, E. J. Hall.

Born 1891, Lynchburg, Va. 1898-1905, Public Schools of Lynchburg, Va., up to the High School. 1905-09, Groff School, New York, N. Y. 1913-15, Asst. Mgr., Seward Dredging Co., and gold-dredging company with offices at 200 Fifth Ave., New York, and operations on Solomon River near Nome, Alaska.

Present position: Student, Columbia School of Mines.

Carl Eberhard Carstens, Golden, Colo.

Proposed by William R. Chedsey, Harry J. Wolf, F. W. Traphagen.

Born 1890, Ackley, Iowa. 1907, Ackley (Iowa) High School. 1913, Lake Forest College; B. A.

Present position: Student, Colorado School of Mines.

Harrison Bacon Fisher, Cleveland, Ohio.

Proposed by Charles H. Fulton, Zay Jeffries, J. Burns Read.

Born 1893, Conneaut, Ohio.

Present position: Student, Case School of Applied Science.

Guy Roche Johnson, Jr., So. Bethlehem, Pa.

Proposed by H. Eckfeldt, Joseph W. Richards, W. G. Matteson.

Born 1894, Longdale, Va. 1908-10, Birmingham High School, Birmingham, Ala. 1910-12, Bethlehem Preparatory School, Bethlehem, Pa. 1913, University of Alabama. 1913, Lehigh University. 1911, Rodman and levelman, working under Charles Hutchinson, Mellen, Wis. 1912, Same position under Milo Long, Tuscaloosa, Ala.

Present position: Student, Lehigh Univ.

Allen Harold Kline, Cleveland, Ohio.

Proposed by A. W. Smith, Zay Jeffries, J. Burns Read.

Born 1893, Youngstown, Ohio. 1908-11, Rayen High School.

Present position: Student, Case School of Applied Science.

Albert S. Konselman, So. Bethlehem, Pa.

Proposed by W. G. Matteson, H. Eckfeldt, Henry S. Drinker.

Born 1894, New York, N. Y. 1912-15, Lehigh Univ. 1913, Lehigh Valley Coal Co. 1914, Geological work under Dr. B. L. Miller.

Present position: Student, Lehigh Univ.

Gotthold Holda Meinzer, Golden, Colo.

Proposed by William R. Chedsey, Harry J. Wolf, F. W. Traphagen.

Born 1890, Heidelberg, Germany. 1907, Ackley (Iowa) High School. 1912, State Univ. of Iowa; B. A.

Present position: Student, Colorado School of Mines.

August John Wiegand, So. Bethlehem, Pa.

Proposed by H. Eckfeldt, W. G. Matteson, Siegfried Fischer.

Born 1890, Philadelphia, Pa. 1903-07, Grad., Roman Catholic High School, Philadelphia, Pa. 1914, Summer, Surveying corps at Hazelton Division of Lehigh Valley Coal Co.

Present position: Student, Lehigh University.

Ernest B. Zimmer, Cleveland, Ohio.

Proposed by Charles H. Fulton, Zay Jeffries, J. Burns Read.

Born 1893, Cleveland, Ohio. 1907-11, St. Ignatius College.

Present position: Student, Case School of Applied Science.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period May 10 to June 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 102, April to June, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of June 10, 1915.

ANDERSON, AXEL E. 588 Du Pont Bldg., Wilmington, Del.
 BADLAM, STEPHEN, Chief. Engr., Pittsburgh Seamless Tube Co., Beaver Falls, Pa.
 BARDWELL, E. S. 23 Smelter Hill, Great Falls, Mont.
 BARNUM, W. EUGENE. 821 E. 4th Street, Tucson, Ariz.
 BIGELOW, BRAXTON. Care British Eastern Auxiliary Hospitals, Belgrade, Serbia.
 BOSQUI, FRANCIS L. 1 London Wall Bldg., London, E. C., England.
 BRADFORD, ROBERT H. 2320 Lake St., San Francisco, Cal.
 BROOKE, LIONEL. Goffs, Cal.

BRUNTON, FREDERIC K.	138 Prospect Ave., Cripple Creek, Colo.
BUSH, H. E.	Taylorville, Cal.
BUTTERS, CHARLES	6272 Cabot Road, Oakland, Cal.
CALLAWAY, S. D.	Care U. S. Zinc Co., Sand Springs, Okla.
CARPENTER, CLARK B.	R. F. D. 7, Girard, Kan.
CASE, BENJAMIN H.	5 Edgehill Ave., Asheville, N. C.
CAVANAGH, JOHN L.	New Glasgow, N. S., Canada.
CHIDESTER, WALTER BARCLAY	Care Mountain Copper Co., Keswick, Cal.
CHOW, KAI CHI, Engr. of Mines	112 Ninkong Lee, West Gate, Shanghai, China.
CHURCH, JOHN A., JR.	111-12th St., Long Island City, N. Y.
CLAGHORN, C. R.	Durham, King Co., Wash.
CLAGHORN, JAMES L.	Grass Valley, Cal.
CLAUDET, H. H.	6 and 7 Coleman St., London, England.
COLVOCORESSES, G. M., Genl. Mgr., Consolidated Arizona Smelting Co.,	Humboldt, Ariz.
CORNELL, R. T.	80 Maiden Lane, New York, N. Y.
COX, W. ROWLAND	120 Broadway, New York, N. Y.
COXE, EDWARD H.	413 W. Cumberland Ave., Knoxville, Tenn.
CROMWELL, ROBERT H.	82 Beaver Street, New York, N. Y.
DANIEL, WILLIAM B.	El Corpus, Dept. of Choluteca, Honduras.
DAVIDSON, HAROLD O.	N. Carpenter Ave., Iron Mountain, Mich.
DAVIS, J. C., Fourth Vice Pres., American Steel Foundries,	1163 McCormick Bldg., Chicago, Ill.
DICKINSON, EDMUND SARGENT	3423 1/2 Lisbon Ave., Milwaukee, Wis.
DICKMAN, R. N.	P. O. Box B, Wequetonsing, Mich.
DORRANCE, CHARLES, JR., Asst. Supt.	Hudson Coal Co., Scranton Pa.
DRESSER, CLARENCE G.	Care Mine La Motte Co., Mine La Motte, Mo.
DRESSER, JOHN A.	327 Victoria Ave., Westmount, Montreal, Canada.
DUNN, THEODORE S.	801 N. County St., Waukegan, Ill.
DURELL, CHARLES T.	3867 Harvard Boul., Los Angeles, Cal.
EILERS, KARL	Room 3410, 120 Broadway, New York, N. Y.
EYOUNG, DJEVAD	Care Lehigh Coal & Navigation Co., Lansford, Pa.
FOSTER, E. LE NEVE	The Perrenoud, Denver, Colo.
FULLER, JOHN T., Min. Engr.	Yardelle, Newton Co., Ark.
GARD, IRVING R.	Care Columbia River Coal Dock Co., North Portland, Ore.
GEIGER, W. F.	P. O. Box 226, Miami, Ariz.
GILL, PHILIP J.	Room 32, 40 Cedar St., New York, N. Y.
GOETTER, FRED B., Mexican Candelaria Co., S. A.,	Room 401, Mechanics Institute Bldg., San Francisco, Cal.
GOODWIN, R. H., 314 Winchester House, Old Broad St., London, E. C., England.	
GOULD, H. J.	121 Pitt St., Sydney, N. S. W., Australia.
GRAPFNER, M. F.	General Delivery, Butte, Mont.
GRIDLEY, HAINES, Mgr.	Oro Fina Mining Co., East Auburn, Cal.
GUGGENHEIM, HARRY FRANK	120 Broadway, New York, N. Y.
HAFF, R. E. T.	duPont Powder Co., City Point, Va.
HALDEMAN, GEORGE T.	16 E. South St., Wilkes-Barre, Pa.
HAMILTON, JOHN W. H.	17 Battery Place, New York, N. Y.
HANAHAN, MARION LOTHROP	28 George St., Charleston, S. C.
HANSELL, N. V.	17 Battery Place, New York, N. Y.
HAYES, C. WILLARD	3432 Ashley Terrace, Washington, D. C.
HEIDENREICH, W. LEE, Mgr.	Georgia Peruvian Ochre Co., Cartersville, Ga.
HEINDL, ALEXANDER J.	Bodaibo, Province of Irkutsk, Siberia.
HIGGINS, EDWIN	Care U. S. Bureau of Mines, Ironwood, Mich.
HOWARD, FERDINAND	105 W. 72d St., New York, N. Y.
INOUE, TADASHIRO	83 Shimo-osaki-machi, Yebara-gori, Tokyo, Japan.
JACKSON, F. H.	Care Mines Co. of America, Mills Bldg., El Paso, Texas.
JONES, W. STRICKLER	120 S. Elberon Ave., Atlantic City, N. J.
JONES, ZECHARIAH	Instructed to hold all mail.
KEENEY, ROBERT M.	Care Magnesium Mfg. Corp., Rumford, Me.
KENNEDY, GEORGE A.	825 Symes Bldg., Denver, Colo.
KINZIE, ROBERT A., Min. Engr., First National Bank Bldg., San Francisco, Cal.	
KOERTING, BERTHOLD	Casilla 1323, Santiago, Chile.
KYNOR, HERBERT D.	622 Quincy Ave., Scranton, Pa.
LABARTHE, JULES	Hobart Bldg., San Francisco, Cal.
LAUHLI, EUGENE HENRI	Room 68, 2 Wall St., New York, N. Y.
LEE, EDWARD CLARENCE	203 Transportation Bldg., Urbana, Ill.

- LEFEVRE, SOLOMON..... Forest Glen, Ulster Co., N. Y.
 LEMMON, DAVID..... West Toledo Mines Co., Kearns Bldg., Salt Lake City, Utah.
 LESLIE, E. H..... *Mining Press*, 300 Fisher Bldg., Chicago, Ill.
 LEWIS, ROBERT S..... Box 902, Stanford Univ., Cal.
 LEWYN, OSWALD MARK..... 146 W. 77th St., New York, N. Y.
 LINDSAY, WILLIAM RUFUS..... Box 955, Juneau, Alaska.
 LINN, WILLIAM J..... Box 774, Tulsa, Okla.
 LONGYEAR, JOHN MUNRO, JR., Care F. L. Barrett, 249 College Ave., Houghton, Mich.
 LONGWELL, A..... 410 Crown Office Bldg., Toronto, Ont., Canada.
 LORING, EDWARD A.... Bewick, Moreing & Co., 62 London Wall, London, E. C.,
 England.
 LORING, WILLIAM J., Bewick, Moreing & Co., 62 London Wall, London, E. C.,
 England.
 LUERSSEN, GEORGE V..... 1558 Perkiomen Ave., Reading, Pa.
 MCGOVERN, R. A., Care Caloric Co., Avenida do Cacs 437, Rio de Janeiro, Brazil.
 MCINTOSH, F. K..... Hurley, Wis.
 MACGREGOR, ALEXANDER GRANT..... Warren, Ariz.
 MACKAY, HENRY S. "The Grange," 167 Denmark Hill, London, S. E., England.
 MAGNUS, BENJAMIN,..... Care H. M. Toch, 320 Fifth Ave., New York, N. Y.
 MARR, ROBERT A..... Union, Ore.
 MARSH, ROBERT, JR., Care M. Guggenheim & Sons, 120 Broadway, New York, N. Y.
 MATHER, W. C., The Cleveland-Cliffs Iron Co., Rockefeller Bldg., Cleveland, Ohio.
 MILLARD, WILLIAM J..... Box 158, Tulsa, Okla.
 MITMAN, CARL WEAVER, U. S. National Museum (Division of
 Mineral Technology), Washington, D. C.
 MONTGOMERY, JOHN C..... 165 Broadway, New York, N. Y.
 MORGAN, GEORGE H..... Care City Engineer's Office, Los Angeles, Cal.
 MORRISON, W. L..... 402 Ridge Ave., Canonsburg, Pa.
 MORSE, PHILIP S..... 157 Walnut St., Brookline, Mass.
 MOSER, GEORGE F..... Oatman, Mohave Co., Ariz.
 MOSES, PERCIVAL SNEED, Care Société Forminière, Kinshasa, Congo Belge,
 West Africa.
 MUTER, A. F..... 421 W. Second St., Los Angeles, Cal.
 NAKAMURA, KEIJIRO, Baron Sumitomo's General Head Office, Kitahama,
 Osaka, Japan.
 NEWBERRY, ANDREW W., Care Col. William Stevenson (Personal), Skagway, Alaska.
 NEUSTAEDTER, A., Genl. Mgr..... Arminius Chemical Co., Mineral, Va.
 NEWELL, FREDERICK H..... University of Illinois, Urbana, Ill.
 NEWELL, G. S..... Asarco, Dgo., Mexico.
 NIGHMAN, C. E..... 26 W. 35th St., Bayonne, N. J.
 NOLD, H. E..... 556 Adams St., Gary, Ind.
 ORR, JOHN F., Sales Engr..... Care Chalmers & Williams, Chicago Heights, Ill.
 OTAGAWA, MASAYUKI..... 57 Haramachi Ichome, Ushigomeku, Tokyo, Japan.
 PALMER, WILLIAM FREDERICK, Assayer and Min. Engr., Calavada Copper Co.,
 Luning, Nev.
 PARK, WALTER E., Eastern Mgr., Chalmers & Williams, Equitable Bldg.,
 New York, N. Y.
 PATCHELL, FRED J..... 4516 N. Lincoln St., Chicago, Ill.
 PERKINS, EDWIN T..... 2102 Kentucky St., Joplin, Mo.
 PERRY, ROBERT S., Kingscourt Apartments, 36th and Chestnut Sts.,
 Philadelphia, Pa.
 PETERS, RICHARD, JR.... Care W. J. Rainey, 52 Vanderbilt Ave., New York, N. Y.
 PETTY, MORRIS K..... Ellsworth, Pa.
 POIRIER, C. HERBERT..... Vipond Mine, Schumacher, Ont., Canada.
 POLHEMUS, J. H., Asst. Mines Mgr., New Jersey Zinc Co., 55 Wall, New York, N. Y.
 PROBERT, FRANK H..... 28 Oak Vale Ave., Berkeley, Cal.
 PYNE, WALTER F., Phi Gamma Delta House, 538 W. 114th St., New York, N. Y.
 RENO, CHARLES A..... Ripley, Nev.
 REYNOLDS, GEORGE B..... 3 Calle de Lago, Maracaibo, Venezuela.
 RODDEWIG, GEORGE W..... Care Federal Lead Co., Flat River, Mo.
 RUEBEL, ERNEST HERTEL..... 4649 Cottage Ave., St. Louis, Mo.
 RYPINSKI, J. E..... Box 194, Moscow, Pa.
 SANDERS, BERNARD H., Itabira Iron Ore Co., Ltd., Itabira de Matto Dentro,
 Minas Geraes, Brazil.
 SCHUETTENHELM, JOHN B..... Aurora, Nev.

- SCHUMACHER, WILHELM...Care Allgemeine Brikettierungs Gesellschaft,
Unter den Linden 8, Berlin, W., Germany.
- SHANKS, DAVID W.....Douglas City, Trinity Co., Cal.
- SHEADLE, J. H., Vice-Pres., Cleveland-Cliffs Iron Co., Rockefeller Bldg.,
Cleveland, Ohio.
- SLATER, J. L., JR.....2933 Lincoln Ave., Alameda, Cal.
- SLEEMAN, H. R.....Palace Hotel, Perth, West. Australia.
- SMITH, CHARLES H.....Care Blair & Co., 24 Broad St., New York, N. Y.
- SMITH, JAY MARK.....Care Ingersoll-Rand Co., Havana, Cuba.
- SMITH, SUMNER S.....U. S. Mine Inspector, Juneau, Alaska.
- STANTON, F. MCM.....15 William Street, New York, N. Y.
- STIEBEL, H. J.....1897 Beacon St., Brookline, Mass.
- TAYLOR, HENRY B.....Bartlesville, Okla.
- THOMSON, S. C.....120 Broadway, New York, N. Y.
- THROPP, J. E., JR.....Alburtis, Pa.
- TURNER, G. D. B.....4 Lloyds Ave., London, E. C., England.
- TUSCHKA, OTTO.....Lavernia, Tex.
- UYENO, KAGEAKI.....43 Honmuracho, Azabu, Tokyo, Japan.
- VAN BARNEVELD, C. E., Mines Dept., Exposition Grounds, Panama Pacific
International Exposition, San Francisco, Cal.
- VAN ZWALUWENBURG, A.....University Club, Mexico City, Mexico.
- WALKER, R. T.....1815 Short St., Berkeley, Cal.
- WALLACE, WILLIAM JOHN...Care Cusi Mining Co., Cusihiurichic, Chih., Mexico.
- WARTENWEILER, FRED, Care Rand Mines, Ltd., Corner House,
Johannesburg, So. Africa.
- WELLS, JAMES S. C.....N. 15th St., Canon City, Colo.
- WHITE, RUSH J., Min. Engr.....519 Bank St., Wallace, Idaho.
- WILLIAMS, J. C.....Box 553, Golden, Colo.
- WILMOT, H. CLIFFORD.....Baker, Ore.
- WILSON, ELWOOD J.....Apartment 66, 540 W. 122d St., New York, N. Y.
- WILSON, FREDERICK C., Prof. of Civ. and Sanitary Engrg.,
Clarkson College of Technology, Potsdam, N. Y.
- WILSON, GEORGE BENNETT.....1002 Main St., Racine, Wis.
- WINCHELL, HORACE V.....826 First National-Soo Line Bldg., Minneapolis, Minn.
- WISHON, W. W.....Duplex Mining Co., Searchlight, Nev.
- WONG, W. A.....Care Tayeh Iron Mines, Tayeh, Hupeh, China.
- WRIGHT, PERCY E., Mech. Engr.....Carbonado, Wash.
- YEATMAN, POPE.....120 Broadway, New York, N. Y.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
BOYS, H. R.....	Aurora Cons. Min. Co., Aurora, Nev.
CLERC, CAMILLE.....	92 Rue Jouffrey, Paris, France.
CHARY, CHARLES N.....	Kimberly, Nev.
EATON, E. R.....	Manatawny Bessemer Ore Co., Manatawny, Pa.
FINLEY, WALTER H.....	Logmont, Ky.
GREY, GEORGE R.....	54 Ameshoff St., Johannesburg, Transvaal, South Africa.
HEBBARD, AUSTIN.....	Box 98, Langlaate, Transvaal, South Africa.
HORCASITAS, A. S.....	Care Tanawah Gold Min. Co., Sweet Water, Nev.
JONES, THOMAS J.....	Perm Govt., via Petrograd, Russia.
KNOERTZER, H.....	56 Rue Balagny, Paris, France.
LANGLEY, SETH S.....	601 W. 190th St., New York, N. Y.
McCARRICK, E.....	628 N. Serrano Ave., Los Angeles, Cal.
McKIM, JOHN W.....	532 Dooly Block, Salt Lake City, Utah.
MAINWARING, H. M. C.....	Chillagoe, Ltd., Chillagoe, Queensland, Australia.
MORRIS, CHARLES E.....	Pony, Madison Co., Mont.
NOBS, FREDERICK W.....	La Leonesa, Matagalpa, Nicaragua.
RALPH, E. W.....	Care Boston Ely Min. Co., Kimberly, Nev.
SALE, A. J.....	Giroux Cons. Mines Co., Kimberly, Nev.
SUNDERHAUF, A. E.....	Paramaribo, Dutch Guiana.
TIMMONS, COLIN.....	Jamestown, Cal.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period May 10 to June 10, 1915:

Date of Election.	Name.	Date of Decease.
1875	*BIRKINBINE, JOHN.....	May 14, 1915
1911	*BRODRICK, C. T.....	May 6, 1915
1914	*FREEMAN, RICHARD RICH, JR.....	May 6, 1915
1906	*GUITERMAN, FRANKLIN.....	May 8, 1915
1876	*HALL, EDWARD J.....	Sept. 17, 1914
1888	*KEIGHLEY, FRED C.....	April 14, 1915
1896	*PEARSON, FRED S.....	May 6, 1915

* Member.

AMERICAN INSTITUTE OF MINING ENGINEERS
EXECUTIVE COMMITTEES OF LOCAL SECTIONS

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New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*, JOHN H. JANEWAY, *Vice-Chairman*.
F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
P. A. MOSMAN, *Treasurer*.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*, ALFRED C. LANE, *Vice-Chairman*.
HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.

FRANK A. ROSS, *Chairman*, RUSH J. WHITE, *Vice-Chairman*.
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.

I. F. LAUCKS, *Chairman*, J. F. MENZIES, *Vice-Chairman*.
GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
H. L. MANLEY.

Southern California

THEODORE B. COMSTOCK, *Chairman*, SEELEY W. MUDD, *Vice-Chairman*.
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
A. B. CARPENTER, C. COLCOCK JONES.

Colorado

CHARLES A. CHASE, *Chairman*, S. A. IONIDES, *Vice-Chairman*.
C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*, JAMES L. BRUCE, *Vice-Chairman*.
DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.

G. HOWELL CLEVINGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
F. W. BRADLEY, ANDREW C. LAWSON.

Pennsylvania Anthracite Section

R. V. NORRIS, *Chairman*.

CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
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BY ARTHUR F. TAGGART,* NEW HAVEN, CONN.

(San Francisco Meeting, September, 1915)

THE following conclusions on the work of the Hardinge mill are based on data furnished to the writer by the Hardinge Conical Mill Co. in the form of the mesh cards hereto appended. Energy units (E. U.) and relative mechanical efficiencies (R. M. E.) are computed by the "volume method" of Stadler.¹ Screen apertures used are the average apertures of testing screens of the meshes given.

Card 122. June 28, 1912. Vipond Porcupine Mines Co., Ltd., Schumacher, Ont., Canada.

Ore from mill bin. Gangue, quartz and basalt.

4.5 ft. by 13 in. ball mill.

Capacity, 48 tons per 24 hr.

Charge, 4,000 lb. balls.

Speed, 33 rev. per minute.

Horsepower, 15 to 17.

Water, 100 per cent. by weight. (50 per cent. ?)

Product deslimed and oversize reground in pebble mill, see Card 113.

Feed to mill through 2-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 1 in.	5.50	
+ ¾ in.	28.00	
+ ½ in.	30.00	
+ ¼ in.	19.72	
+ 10	10.87	2.10
+ 20	2.42	8.00
- 20	3.51	
+ 40		22.68
+ 60		10.50
+ 80		11.80
+ 100		3.90
- 100		40.15

* Sheffield Scientific School.

¹ H. Stadler: Grading Analyses and Their Application, *Transactions of the Institution of Mining and Metallurgy*, vol. xix, p. 471 (1910-11).

Arthur F. Taggart: The Work of Crushing, *Trans.*, xlviii, 153 (1914).

Card 107. Jan. 30, 1912. Miami Copper Co., Miami, Ariz.

Material from mill bin. Gangue, siliceous porphyry.

6 ft. by 16 in. ball mill.

Capacity, 351 tons per 24 hr.

Charge, 4 tons balls.

Speed, 28 rev. per minute.

Horsepower, 35 net.

Water, 50 per cent. (approx.).

Elevation of feed end, 2 in.

Consumption of balls, 0.578 lb. per ton of ore crushed.

Feed to mill through 1.5-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ ¾ in.	6.7	
+ ½ in.	19.7	
+ ¼ in.	28.5	5.2
+ 10	22.3	16.3
+ 20	7.5	20.3
+ 40	3.0	11.9
+ 50	1.1	4.9
+ 60	1.6	6.8
+ 80	0.8	3.4
+100	0.8	2.8
+150	0.9	3.1
+200	1.0	3.5
-200	7.2	21.8

Card 155. Aug. 3, 1914. Britannia Mining & Smelting Co., Britannia Beach, B. C., Canada.

Jig tailing. Gangue quartzose, very hard.

6 ft. by 16 in. ball mill.

Capacity, 251 tons per 24 hr., average of six tests.

Charge, 8,200 lb. of 2-in. cast-iron balls.

Speed, 28 rev. per minute.

Horsepower, 38 to 40.

Water, 40 per cent.

Elevation of feed end, 0

Consumption of balls, 0.72 lb. per ton of ore.

Lining in first-class condition after 3 months' run.

This mill is taking "pebble mill feed" (all through ¼-in. aperture), but using small balls instead of pebbles as a grinding medium. These data are, therefore, not included in the averages in Table II.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	60.4	0.4
+ 20	28.8	5.5
+ 30	5.8	10.9
+ 40	1.1	6.0
+ 60	1.9	18.0
+100	1.2	23.3
-100	1.1	34.2

Card 192. Nov. 17, 1914. McIntyre Porcupine Mines, Schumacher,
Ont., Canada.

Quartz and schist.

6 ft. by 16 in. ball mill.

Capacity, 150 tons per 24 hr.

Charge, 8,000 lb. of balls.

Speed, 28 rev. per minute.

Horsepower, 36.

Water, 50 per cent.

Elevation of feed end, 1.25 in.

Consumption of balls, 0.5 lb. per ton of ore ground.

Feed to mill through 2-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 1.5 in.	6.18	
+ 1 in.	16.62	
+ ½ in.	34.36	
+ ¼ in.	22.90	
+ 10	19.94	0.70
+ 20		5.49
+ 40		18.46
+ 60		11.61
+100		10.83
+200		11.84
-200		41.07

Card 156. June 25, 1914. McIntyre Porcupine Mines, Schumacher,
Ont., Canada.

Quartz and schist, hard.

6 ft. by 16 in. ball mill.

Capacity, 150 tons per 24 hr., average of nine months.

Charge, 4 tons of balls.

Speed, 28 rev. per minute.

Horsepower, 36.

Water, 1.5 tons KCN solution to 1 ton of dry ore.

Elevation of feed end, 1.25 in.

Consumption of balls, 0.5 lb. per ton of ore.

Feed to mill through 2-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 2	73.4	
+ 4	12.2	
+ 10	7.3	
+ 20	2.4	15.6
+ 40	0.5	19.2
+ 60	1.2	
+ 80		15.2
+100	0.6	3.8
+200	0.7	9.0
-200	1.7	37.2

Card 191. Sept. 14, 1914. Buckhorn Mines Co., Buckhorn, Nev.

Soft talcose gold ore. Gangue, decomposed porphyry and basalt.

6 ft. by 16 in. ball mill.

Capacity, 160 tons per 24 hr.

Charge, 8,000 lb. of balls.

Speed, 28 rev. per minute.

Horsepower, 33.25 input to motor.

Water, 4 to 1.

Elevation of feed end, 1.5 in.

Consumption of balls, 0.45 lb. per ton of ore.

Feed to mill through 1.5-in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ ¼ in.	35.2	
+ ⅜ in.	12.2	
+ ½ in.	13.2	
+ ⅝ in.	13.8	
+ 20	12.2	3.0
- 20	13.4	
+ 40		12.0
+ 80		31.0
+100		10.0
+150		16.0
-150		28.0

Card 121. Bunker Hill & Sullivan Mining & Concentrating Co., Kellogg, Idaho.

Middling from 3-mm. jigs and tables. Gangue, quartzite and siderite.

6 ft. by 22 in. pebble mill.

Capacity, 60.3 tons per 24 hr.

Speed, 32 rev. per minute.

Horsepower, 16 (?).

Water, not taken. Previous test gave 75 per cent.

Elevation of feed end, 0.5 in.

Pebble load, 4,000 lb. (?).

Feed to mill through 3-mm. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	3.0	
+ 20	50.7	
+ 40	21.8	
+ 60	3.5	2.8
+ 80	2.5	13.0
+100	4.5	14.1
+150	5.4	15.8
+200	4.9	12.5
-200	3.7	41.8

Card 113. June 28, 1912. Vipond Porcupine Mines Co., Ltd., Schumacher, Ont., Canada.

Oversize of Colbath classifier. Gangue, quartz and basalt.

6 ft. by 72 in. pebble mill.

Capacity, 40 tons per 24 hr.

Charge, 9,000 lb. of pebbles.

Speed, 27 rev. per minute.

Horsepower, 30.

Water, 50 per cent.

Elevation of feed end, 0.

Product deslimed and oversize returned.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	16.64	
+ 20	27.42	
+ 40	27.04	
+ 60	8.88	
+ 80	7.04	0.15
+100	5.00	1.95
+150	5.05	
+200	1.95	29.10
-200		68.80

Card 108. Jan. 30, 1912. Miami Copper Co., Miami, Ariz.

Product from 16 by 42 in. rolls. Gangue, altered schist.

8 ft. by 22 in. pebble mill.

Capacity, 101 tons per 24 hr.

Charge, 10,000 lb. of pebbles.

Speed, 27 rev. per minute.

Horsepower, 36.

Water, 63 per cent.

Elevation of feed end, 1.5 in.

Product desired to pass 30 mesh with a minimum of slime.

Feed through $\frac{1}{2}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 4	10.7	
+ 10	28.8	
+ 20	37.0	3.4
+ 30	11.4	9.2
+ 40	1.8	7.3
+ 60	1.5	13.9
+ 80	0.4	7.8
+100	0.6	7.2
+150	0.5	7.3
+200	0.7	8.0
-200	6.6	35.9

Card 80. Aug. 16, 1911. Miami Copper Co., Miami, Ariz.
 Product from 16 by 42 in. rolls. Gangue, altered schist.
 8 ft. by 22 in. pebble mill.
 Capacity, 180 tons per 24 hr.
 Charge, 10,000 lb. of pebbles.
 Speed, 27 rev. per minute.
 Horsepower, 36.
 Water, 60 to 65 per cent.
 Elevation of feed end, 1.5 in.
 Product desired to pass 30 mesh with a minimum of slime.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 4	0.4	
+ 10	45.0	
+ 20	32.7	2.2
+ 30	8.4	6.8
+ 40	1.7	7.0
+ 60	1.8	15.5
+ 80	0.8	8.9
+100	0.7	7.7
+150	0.8	8.5
+200	0.4	5.3
-200	6.3	38.1

Card 109. Federal Mining & Smelting Co., Wallace, Idaho.
 Middling from jigs. Gangue, quartzite and siderite.
 8 ft. by 22 in. pebble mill.
 Capacity, 110 to 115 tons per 24 hr.
 Charge, 10,000 lb. of pebbles.
 Speed, 28 rev. per minute.
 Horsepower, 35.8 net.
 Water, 60 per cent.
 Elevation of feed end, 0.
 Consumption of pebbles, 2 lb. per ton.
 Silix lining, life 13 months.
 Feed to mill through $\frac{3}{16}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	41.2	
+ 20	45.8	0.1
+ 30	8.0	1.3
+ 40	3.0	6.7
+ 60	1.3	15.6
+ 80	0.3	16.4
+100		14.5
+150	0.3	7.0
+200		13.4
-200	0.1	25.0

Card 75. Sept. 7 to 9, 1911. Federal Mining & Smelting Co., Wallace, Idaho.

Coarse Wilfley middling. Gangue, quartzite and siderite.

8 ft. by 22 in. pebble mill.

Capacity, 99.36 tons per 24 hr.

Charge, 5 tons of pebbles (approx.)

Speed, 28 rev. per minute.

Horsepower, 35.3 net.

Water, 55 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 1.5 to 2 lb. per ton.

Feed to mill through 5-mm. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 20	35.0	
+ 30	31.0	1.0
+ 40	18.5	3.0
+ 60	13.5	7.5
+ 80	1.0	7.0
+100	1.0	10.5
+200		4.0
-200		67.0

Card 136. Federal Mining & Smelting Co., Wallace, Idaho.

Jig middling. Gangue, quartzite.

8 ft. by 22 in. pebble mill.

Capacity, 111.5 tons per 24 hr.

Charge, 5 tons of pebbles.

Speed, 28 rev. per minute.

Horsepower, 35.3.

Water, 71.8 per cent.

Elevation of feed end, 2 in.

Consumption of pebbles, 2 lb. per ton.

Feed to mill through 4 mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	30.1	
+ 20	47.8	
+ 30	12.6	1.0
+ 40	5.1	4.0
+ 60	2.5	12.0
+ 80	2.0	12.5
+100		13.0
+150		6.5
+200		12.0
-200		39.0

Card 150. Sept. 8, 1913. Vieille Montagne Zinc Co., Cumberland, England.

Zinc-lead ore. Gangue, siliceous limestone.

8 ft. by 22 in. pebble mill.

Capacity, 120 tons per 24 hr.

Charge, 3 tons of pebbles.

Speed, 29.5 rev. per minute.

Elevation of feed end, 4 in.

Feed to mill through $\frac{1}{2}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 4	25.5	
+ 6	30.0	
+ 8	34.0	
+ 10	7.5	0.2
+ 20	2.7	4.8
+ 40		18.0
+ 60		17.9
+ 80		15.2
+100		4.0
+200		25.8
-200		13.5

Card 34. Nov. 2, 1910. Calumet & Hecla Mining Co., Lake Linden, Mich.

Tailing from jigs. Gangue, Lake conglomerate.

8 ft. by 22 in. pebble mill.

Capacity 40 to 45 tons per 24 hr.

Charge, 3 tons of pebbles.

Speed, 27 rev. per minute.

Horsepower, 34 to 37.

Water, 40 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 2 lb. per ton of ore.

Feed to mill through $\frac{1}{4}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 6	0.25	
+ 8	2.70	
+ 10	5.10	
+ 20	25.70	0.02
+ 40	36.40	0.15
+ 60	19.35	2.10
+ 80	7.00	6.55
+100	1.40	6.00
+150	1.45	16.90
+200	0.70	26.70
-200	0.20	40.80

Card 33. Nov. 3, 1910. Copper Range Consolidated, Freda, Mich.

Tailing from jigs. Gangue, Lake amygdaloid.

8 ft. by 30 in. pebble mill.

Capacity, 65 tons per 24 hr.

Charge, 5 tons of pebbles.

Speed, 28 rev. per minute.

Water, 64 per cent.

Elevation of feed end, 1 in.

Consumption of pebbles, 2.5 lb. per ton of ore.

Feed to mill through $\frac{1}{4}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 6	10.3	
+ 8	30.0	
+ 10	25.0	
+ 20	29.0	
+ 40	4.5	2.7
+ 60	0.5	8.5
+ 80	0.1	3.3
+100		9.4
+150		30.2
+200		10.9
-200		34.0

Card 142. Oct. 9, 1913. Arizona Copper Co., Morenci, Ariz.

Screened roll product. Gangue, porphyry.

8 ft. by 36 in. pebble mill.

Capacity, 208 tons per 24 hr.

Charge, 10,500 lb. of pebbles.

Speed, 29 rev. per minute.

Horsepower, 55.

Water, 59 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 2.4 lb. per ton.

Feed to mill through $\frac{1}{2}$ -in. mesh.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ $\frac{3}{8}$ in.	7.6	
+ 4	26.0	
+ 8	25.6	
+ 10	12.1	0.6
+ 20	12.0	10.2
+ 30	4.8	13.9
+ 40	2.3	10.0
+ 60	2.0	11.7
+ 80	0.7	5.6
+100	0.4	4.4
+150	0.2	3.5
+200	0.5	5.4
-200	5.8	34.7

TABLE I

No.	Diameter of Mill, Feet	Length of Cylinder, Inches	Type	Mining Company	Gangue	Material	Charge Balls or Pebbles, Pounds
122	4.5	13	Ball....	Vipond Porcupine Mines Co.	Quartz and basalt.....	Ore from mill bin.....	4,000
107	6	16	Ball....	Miami Copper Co.....	Siliceous porphyry.....	Ore from mill bin.....	8,000
155*	6	16	Ball....	Britannia M. & S. Co.	Quartzose, very hard...	Jig tailing.....	8,200
192	6	16	Ball....	McIntyre Porcupine Mines.	Quartz and schist.....	Rock-crusher product...	8,000
156	6	16	Ball....	McIntyre Porcupine Mines.	Quartz and schist.....	Rock-crusher product...	8,000
191	6	16	Ball....	Buckhorn Mines Co....	Decomposed porphyry and basalt.	Rock-crusher product...	8,000
121	6	22	Pebble..	Bunker Hill & Sullivan.	Quartzite and siderite....	Middling from jigs and tables.	4,000(?)
113	6	72	Pebble..	Vipond Porcupine Mines	Quartz and basalt.....	Overize Colbath classifier.	9,000
108	8	22	Pebble..	Miami Copper Co.....	Altered schist.....	Product 16 by 42 in. rolls.	10,000
80	8	22	Pebble..	Miami Copper Co.....	Altered schist.....	Product 16 by 42 in. rolls.	10,000
109	8	22	Pebble..	Federal M. & S. Co.....	Quartzite and siderite....	Jig middling.....	10,000
75	8	22	Pebble..	Federal M. & S. Co.....	Quartzite and siderite....	Coarse Wilfley middling	10,000
136	8	22	Pebble..	Federal M. & S. Co.....	Quartzite and siderite....	Jig middling.....	10,000
150	8	22	Pebble..	Vieille Montagne Zinc Co.	Siliceous limestone.....	Jig middling.....	6,000
34	8	22	Pebble..	Calumet & Hecla.....	Conglomerate.....	Jig tailing.....	6,000
33	8	30	Pebble..	Copper Range Consol...	Amygdaloid.....	Jig tailing.....	10,000
142	8	36	Pebble..	Arizona Copper Co.....	Porphyry.....	Screened roll product...	10,500
135	7	144	Tube...	Federal M. & S. Co.....	Quartzite.....	Wilfley middling.....	18,000

* See note on mesh card. Note R. M. E.

Card 135. Oct. 26, 1912. Federal Mining & Smelting Co., Morning, Idaho.

Wilfley middling. Gangue, quartzite.

7 ft. by 12 ft. tube mill.

Capacity, 124 tons per 24 hr.

Charge, 9 tons of pebbles.

Speed, 22.25 rev. per minute.

Horsepower, 86.

Water, 58.8 per cent.

Elevation of feed end, 0.

Consumption of pebbles, 4 to 5 lb. per ton of ore.

Mesh	Feed, Per Cent.	Discharge, Per Cent.
+ 10	1.5	
+ 20	11.5	
+ 30	23.5	0.5
+ 40	23.0	2.0
+ 60	23.5	7.0
+ 80	10.0	11.0
+100	4.5	15.0
+150	1.0	5.5
+200	1.0	14.5
-200	0.5	44.5

TABLE I—Continued

Speed, Rev. per Min.	Tons per 24 Hr.	Horsepower	Tons per Horsepower	Feed		Energy Units	Discharge		Per cent. — 200 Mesh.	Energy Units	Difference, E.U.	Relative Mech. Effic. (R.M.E.)	Per Cent. Water in Feed	Elevation of Feed End, Inches	Pebbles or Balls Pounds per Ton
				All pass, Mm.	Average Size, Mm.		All Pass, Mm.	Average Size, Mm.							
33	48	16	3.0	50.8	12.01	280.2	6.3	0.26	40.2 ^b	1,423.4	1,143.2	34.3	50		
28	351	35	10.0	38.1	6.01	665.3	12.7	0.86	21.8	1,247.0	581.7	58.3	50	2	0.578
28	251	39	6.4	6.35	1.28	912.5	6.35	0.24	34.2 ^b	1,432.7	520.2	33.5	40	0	0.72
28	150	36	4.2	50.8	12.7	313.1	6.35	0.19	41.07	1,705.1	1,392.0	58.0	50	1.25	0.5
28	150	36	4.2	50.8	10.04	182.5	1.65	0.25	37.2	1,330.0	1,147.5	47.8	60	1.25	0.5
28	160	33.25	4.8	38.1	7.09	525.4	3.2	0.16	28.0 ^c	1,535.6	1,010.2	48.6	80	1.5	0.45
32	60.3	16(?)	5.2	3.0	0.58	1,092.8	0.36	0.09	41.8	1,686.9	594.1	30.9	75(?)	0.5	
27	40	30	1.3	3.0(?)	0.65	1,154.6	0.24	0.05	68.8	1,823.4	668.8	9.19	50	0.0	
27	101	36	2.8	25.4	1.36	973.8	1.65	0.19	35.9	1,573.2	599.4	16.8	63	1.5	
27	180	36	5.0	12.7	1.1	990.3	1.65	0.17	38.1	1,591.3	601.0	30.0	62.5	1.5	
28	112.5	35.8	3.1	4.7	1.12	943.2	1.65	0.14	25.0	1,594.7	651.5	20.5	60	0.0	2.0
28	99.36	35.3	2.8	5.0	0.56	1,040.9	0.83	0.09	67.0	1,750.6	709.7	20.0	55	0.0	1.5-2
28	111.5	35.3	3.3	4.7	0.99	986.4	0.83	0.12	39.0	1,649.4	663.0	21.7	71.8	2.0	2.0
29.5	120	35(?)	3.4	12.7	3.15	632.5	2.4	0.21	13.5	1,492.3	859.8	29.5		4.0	
27	42.5	35.5	1.2	6.35	0.56	1,195.6	1.65	0.08	40.8	1,614.7	419.1	5.0	40	0.0	2.0
28	65	46(?)	1.4	6.35	1.72	825.8	0.83	0.10	34.0	1,652.9	827.1	11.7	64	1.0	2.5
29	208	55	3.8	12.7	2.89	786.5	2.36	0.27	34.7	1,502.1	715.6	27.1	59	0.0	2.4
22.25	124	86	1.4	2.3	0.41	1,051.4	0.83	0.10	44.5	1,690.2	638.8	9.2	58.8	0.0	4-5

^b — 100 mesh. ^c — 150 mesh.

Table I is a summary of the data presented on the mesh cards. Table II is taken from Table I.

Apparently the ball mill does more work per unit of power input than the pebble mill, while the pebble mill is, in the same way, more efficient than the tube mill in the one instance cited. Nos. 107 and 142 show the effect of overloading the mill. The relative mechanical efficiencies in these two cases are raised above the average at the expense of the character of the product.

TABLE II

	6 ft. by 16 in. Ball Mill	8 ft. by 22 in. Pebble Mill
Average maximum size of feed, mm.....	44.5	9.7
Average size of feed, mm.....	9.0	1.26
Average maximum size of discharge, mm.....	6.0	1.5
Average size of product, mm.....	0.37	0.14
Average per cent. of -200 mesh in discharge.....	28.9*	37.0
Average per cent. of -200 mesh in discharge, no slope.....		44.3
Average per cent. of -200 mesh in discharge, 0.5 to 4 in. slope.....		31.6
Reduction ratio, range.....	7 to 67	6 to 15
Reduction ratio, average.....	39.6	8
Average size of discharge, no slope, mm.....		0.10
Average size of discharge, slope 0.5 to 4 in.....		0.17
Average tonnage.....	203	110
Average tonnage at no slope.....		85
Average tonnage at 0.5 to 4 in. slope.....		128
Average horsepower.....	35.06	35.6
Average charge, balls or pebbles, tons.....	4	4.5
Average ball or pebble consumption, pounds per ton.....	0.51	1.94
Average relative mechanical efficiency.....	53.2	20.5
Average percentage of water in feed.....	60	58.7
Average revolutions per minute.....	28	27.8

* Nos. 155 and 191 estimated.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Additional Data on Origin of Lateritic Iron Ores of Eastern Cuba

BY C. K. LEITH,* PH.D., AND W. J. MEAD,† M.A., MADISON, WIS.

(San Francisco Meeting, September, 1915)

In 1911, we published in the *Transactions* a brief account of the lateritic alterations of serpentine in eastern Cuba, producing the important iron-ore deposits of the Mayari and Moa districts.¹ The special feature of that article was a quantitative treatment of the alterations, based on analyses, to show just what had happened in terms of chemical, mineralogical, and physical changes. Professor Kemp has published an interesting account of these ores, containing additional quantitative observations of his own, tending essentially to confirm the nature of the changes that had been worked out.² His discussion suggests to us the desirability of publishing certain additional quantitative data which were not included in our previous article. This seems especially desirable because quantitative measurements of lateritic alterations are rare, and this particular case may be regarded as typical of a fairly wide range of alterations of this kind.

At the time of our examination of Mayari and Moa deposits 29 complete analyses were supplied us by the chemists of the Spanish-American Iron Co., representing a graded series from the unaltered serpentine rock below to the ore at the surface. These analyses are given in the accompanying table.

The mineralogical composition of the ores and rocks is comparatively simple and can be easily calculated from the analyses. In Fig. 1 the mineralogical changes from the serpentine rock below to the lateritic iron ore above, based on the analyses, are expressed in terms of weight, assuming alumina to have remained constant during the alteration.

It is not easy by mere inspection of this table of analyses to determine what the progressive chemical changes have been in development of the ore. In order to bring this out clearly, the analyses have been graphically represented in Fig. 2 in such a fashion as to show the progressive additions and losses of substances during alteration.

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¹ Leith, C. K., and Mead, W. J.: Origin of the Iron Ores of Central and Northeastern Cuba, *Trans.*, xlii, 90 to 102 (1911).

² Kemp, J. F.: The Mayari Iron-Ore Deposits, Cuba, *Bulletin* No. 98, February, 1915, pp. 129 to 154.

*Chemical Analyses Showing Alteration of Serpentine Rock to Iron Ore in
the Mayari District, Cuba*

Analyses supplied by Spanish-American Iron Co.

Depth, Feet	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	Fe	MgO	Cr.	Ni+Co	P.	S.	H ₂ O+	Total
0-1	2.58	15.71	66.20	46.37		0.92	0.38	0.016	0.12	10.20	96.126
1-2	2.38	20.81	64.70	45.34		0.96	0.33	0.022	0.12	10.63	99.952
2-3	1.60	17.43	68.40	47.81		0.96	0.42	0.018	0.14	9.15	98.118
3-4	1.42	14.23	68.70	48.09		1.04	0.36	0.019	0.16	9.50	95.429
4-5	1.56	8.47	70.60	49.46		1.27	0.61	0.016	0.17	10.14	92.836
5-6	2.90	10.24	72.35	50.56		1.66	0.84	0.016	0.20	10.96	99.166
6-7	2.20	8.29	72.90	51.00		2.19	1.09	0.007	0.19	11.35	98.217
7-8	2.68	4.92	71.85	50.28		2.19	1.15	0.006	0.14	11.57	94.506
8-9	3.30	7.25	71.55	50.15		2.39	1.14	0.006	0.16	12.12	97.916
9-10	2.44	6.91	72.40	50.63		2.08	1.21	0.005	0.16	12.35	97.555
10-11	2.42	6.31	71.40	49.94		2.00	1.36	0.005	0.14	12.40	96.035
11-12	2.72	7.05	70.55	49.46		2.08	1.31	0.004	0.15	12.40	96.264
12-13	2.56	6.77	70.20	40.08		1.62	1.37	0.004	0.10	13.50	96.124
13-14	2.52	6.23	70.55	49.46		1.85	1.41	0.005	0.14	13.12	95.825
14-15	2.76	6.58	71.85	50.22		1.89	1.38	0.007	0.21	12.45	97.127
15-16	2.78	6.53	70.00	48.98		2.16	1.33	0.007	0.19	12.35	95.347
16-17	2.98	6.43	69.80	48.84		2.19	1.42	0.007	0.19	12.57	95.587
17-18	3.20	5.53	70.45	49.32		2.00	1.35	0.007	0.15	12.90	95.587
18-19	3.66	6.51	69.20	48.42		2.43	1.34	0.005	0.06	12.73	95.935
19-20	6.84	8.49	63.35	44.32		2.51	1.36	0.004	0.08	12.80	95.434
20-21	7.44	5.13	66.55	46.58		2.27	1.57	0.003	0.09	12.45	95.503
21-22	8.46	4.99	57.80	40.47	0.00	2.16	1.47	0.006	0.08	12.71	87.676
22-23	11.04	8.38	62.10	43.49	0.00	1.85	1.74	0.002	0.09	14.07	99.272
23-24	15.86	4.70	63.90	44.62	0.00	2.19	1.57	0.003	0.09	11.73	100.043
24-25	17.40	4.00	62.90	40.00	0.50	1.85	1.43	0.003	0.12	11.64	99.843
25-26	22.54	4.57	50.25	35.12	6.49	1.89	1.80	0.002	0.06	13.65	101.252
26-27	28.60	4.18	32.85	23.00	18.23	1.12	1.43	0.003	0.09	13.45	99.953
27-28	35.64	2.33	18.25	12.78	27.35	0.77	1.35	0.001	0.06	14.23	99.981
28-29	39.80	1.39	10.14	7.10	33.69	0.26	0.97	0.001	0.06	13.31	99.561

By means of the "straight-line diagram,"³ each analysis has been compared in turn with the analysis of the unaltered serpentine rock. Each point platted on the horizontal scale, for a given constituent, represents the number of grams of altered rock or ore necessary to contain the same amount of that constituent as 100 grams of unaltered serpentine rock. For example, only 7 g. of ore at the surface are required to contain the same amount of alumina as 100 g. of serpentine rock, and hence, if alumina has remained constant, 100 g. of serpentine rock produce only 7 g. of ore at the surface. If any particular constituent is assumed to have remained constant, all constituents platted to the left have been relatively increased and those falling to the right have suffered loss.

Starting with the unaltered serpentine rock below, it is seen that magnesia and silica are very rapidly lost, magnesia practically entirely disappearing at a distance of 5 ft. above the serpentine rock. Silica is reduced to a small minimum above 10 ft. above the serpentine rock and

³ For a detailed description of the straight-line diagram, see *Economic Geology*, vol. vii, No. 2, pp. 141 to 144 (Mar., 1912).

continues essentially constant to the surface. Alumina, iron, and chromium rapidly increase in percentage with the loss of the silica and magnesia. Iron reaches constant maximum percentage at about the same depth that silica reaches its minimum percentage. At a depth of about 10 ft. from the surface the alumina curve turns to the left, showing an increased rate of concentration, while the iron and chromium curves turn to the right. The relative increase of alumina as compared with iron and chromium is believed to be due to reduction and solution of iron in the upper portion of the ore and its transportation downward,

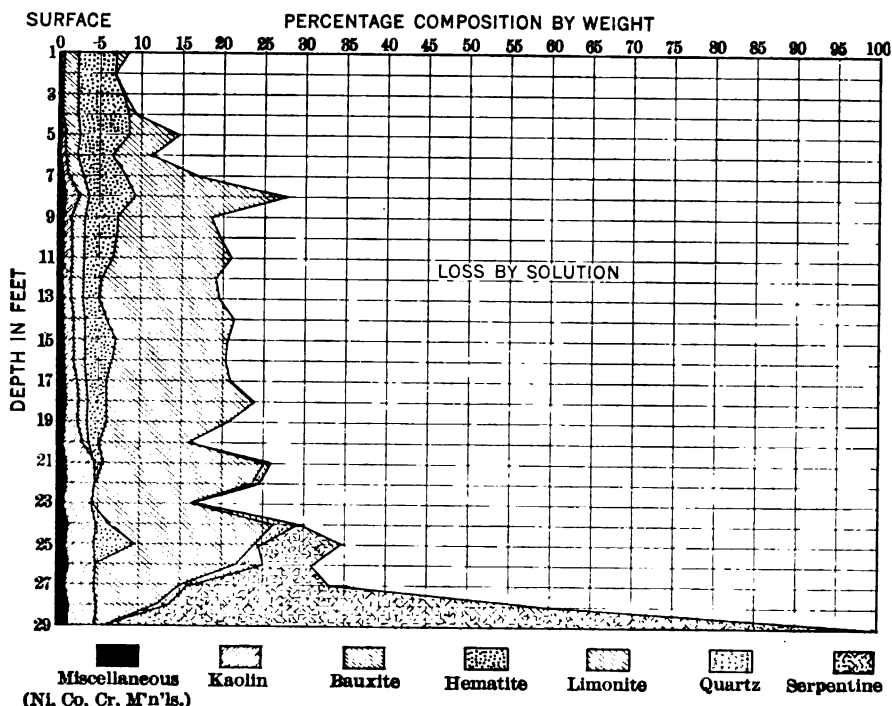


FIG. 1.—PLAT OF MINERAL CHANGES (BASED ON ANALYSES) IN THE LATERITIC ALTERATIONS OF SERPENTINE, PRODUCING THE IRON ORES OF THE MAYARI DISTRICT OF EASTERN CUBA.

resulting in a downward secondary concentration of iron oxide. Nickel and cobalt are lost relative to iron and alumina, the loss being sharply accelerated at about the same depth at which a sharp acceleration in the relative increase of alumina occurs. The percentage of combined water is comparatively constant but in actual amount combined water is lost relative to iron and alumina.

The marked loss of nickel and cobalt during the formation of the ore suggests the possibility of secondary downward concentration of these metals. Descriptions of the nickel ores of New Caledonia indicate that

they are the result of alteration of peridotitic and pyroxenic rocks and of serpentine rocks derived from them, from which the nickel is leached during weathering and carried down and deposited as hydrated silicates

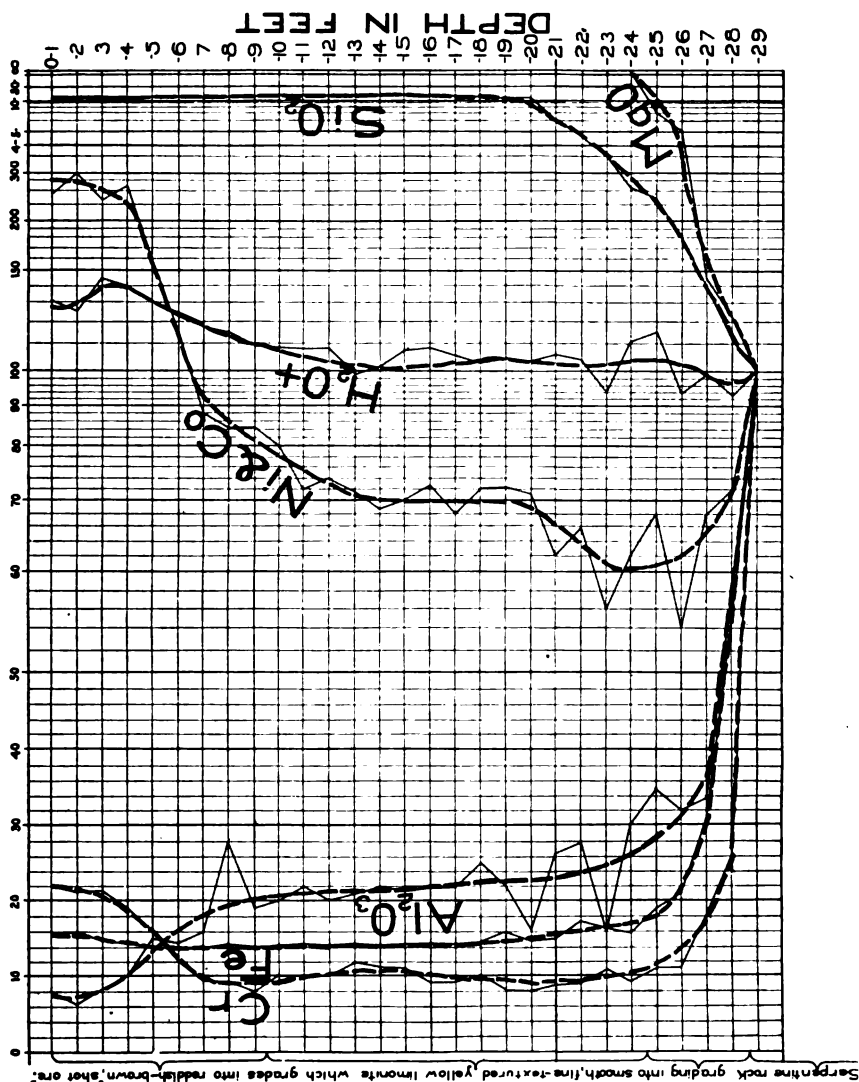


FIG. 2.—PLAT OF CHEMICAL CHANGES IN THE LATERITIC ALTERATION OF SERPENTINE PRODUCING THE IRON ORES OF THE MAYARI DISTRICT OF EASTERN CUBA.

in veins in the underlying unaltered or slightly altered rock. This suggests the possibility of similar occurrence of secondary nickel-bearing minerals in the serpentine rock beneath the Cuban iron-ore deposits.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Notes on Homestake Metallurgy

BY ALLAN J. CLARK,* E. M., LEAD, S. D.

(San Francisco Meeting, September, 1915)

It is nearly three years since the metallurgy of the Homestake ore was discussed with considerable thoroughness, in a paper¹ read before the Institution of Mining and Metallurgy.

Certain changes have been made in this period which are perhaps of sufficient interest to justify a brief description, and it is chiefly with such details and miscellanies that this paper will deal. In the circumstances, a certain amount of repetition is inevitable, but matter treated of in the former paper will be referred to here only when it is necessary for the sake of coherency. For full descriptions of equipment and technique, the former paper should be consulted.

The mineralized slates and schists which constitute the ore vary considerably in composition, but the unoxidized ore, perhaps constituting the major part of the reserves, contains either chlorite or hornblende (cummingtonite), with quartz, carbonates of lime, magnesia and iron, and arsenopyrite, pyrite, and pyrrhotite. Ferrous minerals predominate, and this fact has been an important factor in determining the metallurgical treatment.

With one or two exceptions the minerals noted are of relatively high specific gravity, and the ore as a whole is exceptionally heavy, many determinations giving an average specific gravity of 3.00.

This high specific gravity presents one decided advantage, when the cost of treatment is compared with operations elsewhere, in that the ton, almost universally the basic unit, represents a volume probably less by 10 per cent. than that of many gold ores. On the other hand, this high gravity renders more difficult the discharge of pulp from mortars, its distribution on amalgam tables, and its transportation in launders.

In the same way, as attesting the usual balance of such variations of different ores, it may be noted that the needle-like fibers of hornblende (cummingtonite), interlacing throughout the mass, render the ore more difficult to crush than would be anticipated, but on the other hand assist

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¹ Clark and Sharwood: The Metallurgy of the Homestake Ore, *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 68 (1912-13).

in maintaining free leaching in both sand and slime treatment, retaining their characteristic form as far as they can be traced with the microscope.

The metallurgical equipment consists of:

At the South Side:

- 3 stamp mills (660 stamps) with 36,356 sq. ft. of amalgam plates.
- 1 regrounding plant, with independent cone system and 540 sq. ft. of amalgam plates.
- 4 batteries of cones for classification.
- 3 clarifying-tank houses.
- 1 sand plant.

At the North Side:

- 2 stamp mills (360 stamps).
- 2 tank houses.
- 2 cone houses.
- 1 sand plant.

At Deadwood:

- 1 slime plant, treating the combined slime from South and North sides.

A flow sheet is given in Fig. 1.

The ore supplied to the North Side mills is usually drawn from the upper levels of the mine, and is at least partly oxidized. It is more easily penetrated by cyanide solutions than is the unoxidized ore, and satisfactory extractions are at present made from the sandy portion of the mill tailing without further reduction in tube mills. In almost all other respects, practice at the two divisions of the works is identical, and unless otherwise stated further description will be of operations at the southern branch.

STAMP MILLING

The stamps, when newly shod, weigh 900 lb. and drop 10 in., making 88 drops per minute. Pulp discharges through No. 8 diagonal needle-slot screens. Inside amalgamation is practiced, quicksilver being fed to the mortar, and a copper chuck-block, about $5\frac{1}{2}$ in. wide, being placed inside the mortar, below the discharge screen. To facilitate discharge and distribution of the pulp over the amalgam plates, water is liberally used, the usual ratio being from 10 to 11 parts, by weight, to one of solids.

Electric drive, now in use for more than two years, permits more nearly continuous operation than formerly. A 25-hp. back-gearred motor drives each 10 stamps, transmitting motion by a 16-in. belt.

Chuck-blocks are cleaned in rotation, the usual interval between cleanings being about two weeks; outside plates, except those of the fourth row, are cleaned and dressed daily. Fourth-row plates are dressed at intervals of two days. At the foot of the rows of plates are traps, from which the sands are removed every second or third day. These trap sands are run over a special silvered amalgam plate, the tailing rejoining the main stream of tailing. It was formerly the practice to clean these traps

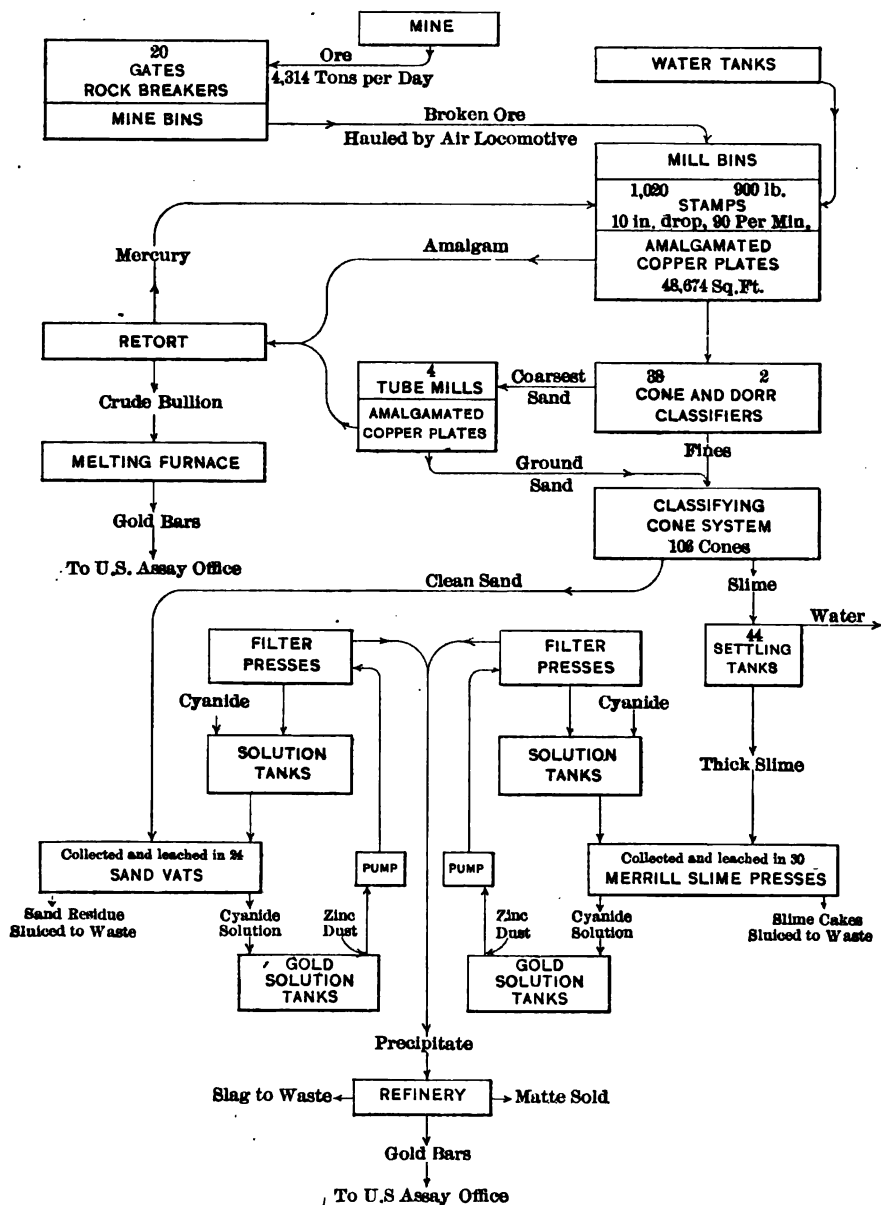


FIG. 1.—FLOW SHEET SHOWING ORE TREATMENT AT HOMESTAKE MINE.

daily, but this is not necessary since the development of the regrinding plant, which acts, in effect, as a large trap for all the mills.

Amalgam is retorted three times each month. Oil-fired retorts are used, the resulting bullion being melted in coke-fired furnaces. The loss of quicksilver in retorting is almost *nil*; the subsequent melting loss, which averages slightly more than 1 per cent. of the weight of the crude bullion, includes some quicksilver not driven off in the retorts.

A new mortar (Fig. 2) has recently replaced the familiar design. In this newer design the old "inside lines," remarkably effective in main-

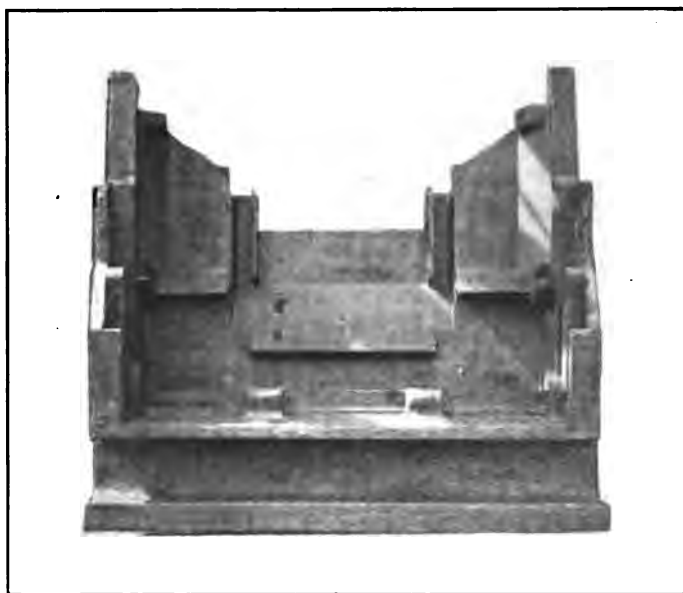


FIG. 2.—NEW OPEN-FRONT MORTAR IN USE AT HOMESTAKE MILLS.

taining a rapid discharge, have been retained. All the front above the screen rest has been cut away, as has also a considerable portion of the back. This has given greater accessibility, with resulting safety to the men when engaged in changing iron and other necessary operations; it permits the use of longer bossheads and simplifies the care of feeders, since an observer at the front of the mortar can determine whether the ore is feeding properly. The slight tendency to splash at the back is corrected by a false back, or apron, of canvas; a small chip tray of 4-mesh screen, resting above the discharge screen, impounds floating particles of wood, which are removed from time to time, washed, and burned. The ashes, which assay as high as \$300 per ton, are set aside to accumulate in sufficient quantity and are eventually sold.

The unoxidized ore, although carrying large quantities of sulphides,

amalgamates well, but not so freely as old records show to have been the case in earlier years, when less of this class of ore came to the mills. A longer time is apparently required to properly incorporate the quicksilver with the pulp, and the resulting amalgam is recovered somewhat farther from the battery. The results given in Table I illustrate this tendency.

TABLE I.—*Comparison of Amalgamation Results on Ores of Different Percentages of Sulphides*

Column I. All mills, three-month period in 1910.

Column II. One mill, crushing unoxidized ore exclusively, three months, 1915.

Of the Total Recovery by Amalgamation, there was Recovered from:	I. Per Cent.	Cumulative Per Cent.	II. Per Cent.	Cumulative Per Cent.
Batteries.....	49.4	49.4	35.6	35.6
First-row plates.....	37.5	86.9	42.9	78.5
Second-row plates.....	4.4	91.3	8.1	86.6
Third-row plates.....	2.3	93.6	3.5	90.1
Fourth-row plates*.....	0.9	94.5	2.0	92.1
Skimmings.....	4.0	98.5	6.9	99.0
Trap sands.....	1.5	100.0	1.0	100.0
Percentage of ore value.....		71.7		69.7

* Only two mills (440 stamps) equipped with a fourth row of plates.

The "skimmings" of the above table are recovered from re-treatment of the foul amalgam, sulphide particles, etc., removed from the cleanup sink during the cleanup of the chuck-block and plate amalgam.

These cleanings are re-treated in a small barrel 24 in. in diameter by 302 in. long, in which iron balls or pebbles are placed, the mineral particles being ground and the amalgam recovered. The rejected sulphides, carrying perhaps \$1,000 per ton in gold, are briquetted with water glass and charged to the blast furnace, when smelting cyanide precipitate or by-products.

The increased proportion of the amalgam recovered from the skimmings during the second period under review is a natural result of the increased percentage of sulphide particles in the ore. The item is, of course, more properly a credit to the earlier items, presumably largely to the battery and first-row items. The tendency of fine sulphides to collect in the slight depressions in the surface of an amalgamated plate is well illustrated at the regrinding mill, where it is necessary to wash such a film from the surface several times each day.

It is perhaps not customary to report inquiries which have failed of definite conclusions. The following notes of such an investigation are not without interest, despite the lack of decisive result. Records of all six mills were tabulated over a period of one year, in an effort to determine

what relation, if any, existed between the amount of quicksilver fed to the batteries, the amount of bullion recovered, and the grade of the ore crushed. As was to be expected, individual determinations varied considerably from the average, but for four of the mills, and these fortunately including those crushing ore of the two extremes in value, the points coincide reasonably well with a curve represented by the equation

$$T = \frac{8.5 B - H}{H}$$

when

T = Value of ore (dollars per ton).

H = Troy ounces of quicksilver fed to batteries.

B = Troy ounces crude bullion (\$16 per ounce) recovered.

Of the two remaining mills, one was crushing surface ores with comparatively coarse particles of free gold. This, not unreasonably, showed values for H about 10 per cent. below the values indicated by the equation. The sixth mill gave results nearly 20 per cent. higher than might have been expected. This mill, on account of an insufficient supply of water, crushed nearly 10 per cent. less ore than the average and it is conceivable that the longer retention of the material in the battery caused undue flouring of the quicksilver. It is more probable, however, that the explanation is to be found in the personal equation of the millman in charge.

The crushing units are not in conformity with modern ideas, and there is no doubt that, were the plants to be built anew, radical changes in design would be made. The Homestake, unlike many younger mines, has developed gradually from comparatively small beginnings. The mills represent the growth of 30 years. Built before the cyanide process was known, they then represented advanced milling practice. It must be conceded that the stamp duty of $4\frac{1}{4}$ tons, with 80 per cent. of the tailing passing a 100-mesh screen and 60 per cent. passing a 200-mesh screen, is fairly good even when compared with the results reported from many newer installations.

We find that heavier stamps crush more rock, with little or no change in the sizing of the tailing unless more open screens are used; in which case, even without increasing the falling weight, tonnage may be gained at the sacrifice of sizing. To us it appears that compactness of plant, with the saving both in first cost and, later, in labor and supervision, constitutes the leading claim of the heavy stamp for preferment. Indeed, if the gold metallurgist is ready to dispense with amalgamation—and we are by no means prepared to do this—it is more than probable that he will do well to investigate the crushing practice of modern copper mills before he commits himself to the heavy gravity stamp. Alaska, at the moment, holds more of interest than does Africa.

It may be pertinent to note that I have never encountered a Home-

stake mill tailing, no matter how far advanced in secondary treatment, from which some free gold could not be recovered by laboratory amalgamation tests.

Sundry operating and cost data of stamp milling are appended:

Stamp Mills: Analysis of Lost Time

NOTE.—The time devoted to dressing plates, about 10 battery-minutes daily, is in part included, as whenever possible the work on the battery is done during this period of plate dressing. Data of 660 stamps for 59 consecutive days.

Reason for Loss of Time.	Battery Hours.	
Installing new screens.....	69 : 20	
Installing new shoes, dies, heads.....	194 : 40	
Installing new guides, guide castings.....	10 : 35	
Installing new mortars.....	63 : 55	
Installing new tappets, cams.....	20 : 00	
Installing new cam shafts.....	6 : 25	
Installing other equipment.....	37 : 00	401 : 55
Setting tappets.....	685 : 10	
Pullouts of stems.....	122 : 20	
Broken stems.....	343 : 05	
Broken shoes and dies.....	15 : 10	
Loose cams.....	39 : 50	
Miscellaneous repairs.....	120 : 05	1,325 : 40
Motor trouble.....	253 : 20	
Drive troubles.....	5 : 20	258 : 40
Total lost time.....		1,986 : 15
Total battery hours.....		186,912
Percentage of total time lost.....		1.07

Cost of Stamp Milling

	Stamp- ing	Amalgamating		Total
		Normal Charges	Rebuilding Plate House	
Operating labor.....	\$0.0895	\$0.0164		\$0.1059
Other labor.....	0.0033		\$0.0064	0.0097
Power.....	0.0457			0.0457
Machinery.....	0.0628			0.0628
Water.....	0.0315			0.0315
Sundry supplies.....	0.0097	0.0036 quicksilver 0.0042 silver plating 0.0003 miscellaneous	0.0059 silver plating 0.0018 lumber	0.0255
Total.....	\$0.2425	\$0.0245	\$0.0141	\$0.2811

Power at \$30 per horsepower year; quicksilver at 50c. per pound; castings, 2½c. per pound; mill labor, \$2.97 per 8-hr. shift.

REGRINDING

Operations at the regrinding plant offer little of interest, the practice conforming closely to the usual methods.

The feed to the tube mills is already so fine (only 25 per cent. remains on a 50-mesh screen) that the efficiency of the mills is low. The critical size of sand delivered to the sand plants is about 100 mesh for unoxidized ore, and 80 mesh for oxidized ore. It is advantageous to operate the mills in closed circuit, but it is difficult to do this without permitting sulphide particles which have been sufficiently reduced in size, to remain in the circuit. By introducing a double baffle or trough classifier, containing a hydraulic device, into the slime-overflow end of a Dorr classifier this difficulty has been in a measure overcome, and two of the mills are at present using this system. The accompanying table of tube-mill data has been compiled from earlier records, when none of the tube-mill discharge was returned for further grinding.

Each mill discharges its tailing over an amalgam table, and about \$0.40 is recovered on these per ton ground.

Regrinding Plant Data

Mill Number	1	2	3	4
Manufacturer.....	Denver Engr. Works.	Allis-Chalmers Co.	Allis-Chalmers Co.	Hardinge Conical Mill Co.
Dimensions, feet.....	5 by 14	5 by 18	5 by 18	6 by 6
Speed, rev. per min.....	27.0	28.5	28.0	26.5
Actual horsepower (motor input)...	32.0	42.0	42.0	27.0
Fed from.....	Dewatering cone.	Dorr classifier	Dorr classifier	Dewatering cone.
Tons fed per day.....	98.0	138.0	133.0	93.0
Pebbles per ton.....	1.40	1.24	1.38	1.71
Tons to pass 100 mesh.....	41.0	52.0	49.0	34.0
Tons to pass 100 mesh per horsepower.	1.28	1.24	1.17	1.26

Regrinding Plant: Operating Costs for 1914

	Cost per Ton Fed to Tube Mills	Cost per Ton Reduced to Pass 100-mesh Sieve
Labor.....	\$0.0496	\$0.1289
Pebbles and liners.....	0.0259	0.0673
Renewals, mills and cones.....	0.0074	0.0193
Machine-shop service.....	0.0037	0.0096
Silver plating.....	0.0077	0.0200
Sundries.....	0.0029	0.0075
Power.....	0.0292	0.0759
Total.....	\$0.1264	\$0.3285

Tons fed to mills, 152,106. Tons ground to pass 100-mesh sieve, 58,605. Cost of regrinding, per ton crushed in stamp mills, \$0.0121.

SAND TREATMENT

The sand is leached with cyanide solution in vats 44 ft. in diameter by 9 ft. deep, holding 610 tons of sand each. The operations are distinguished by unusual care in the preparation of the material for extraction, rather than in the extraction itself.

Classification and aeration are the two essentials to successful work. The latter, necessary to overcome the tendency of the ferrous compounds in the ore to remove the vital oxygen from the solutions, is achieved by forcing, at intervals, air under slight pressure into the false bottom, below the filter canvas. The sand charge is suitably drained before this air is applied; the pressure is so adjusted that the air is forced into the charge, yet is so low that the column of sand is nowhere broken or disturbed. To secure this immunity thorough classification is essential, and this has always been recognized as a matter of first importance.

Four batteries of cones, each fed from the discharge of the preceding set, the first three sets acting by gravity alone, the last assisted by a hydraulic connection, deliver a very clean sand to the vats. As a further precaution, the vat is filled with water before sand is turned into it, so that a constant overflow is maintained during filling and a final separation of slime particles added to the sum of the cone separations. As a matter of fact, the sand is so clean when it enters the vat that this final step is not one of strict necessity, but we have found that a charge filled in this manner is in a less compacted condition, presumably easier to leach and certainly easier to discharge.

The action of the air is interesting. Laboratory tests indicate that, on an average, about 75 cu. ft. of oxygen is absorbed by a ton of ore, before its reducing action is corrected. If this air is not supplied from some extraneous source, the solutions are vitiated and extraction ceases. When air is applied and is followed by a water wash, calcium thiosulphate appears in the effluent solutions; when the air is followed by cyanide, the effluents contain sulphocyanides and free cyanide appears only after some time. After a limited time of leaching, the extractive power of the solution decreases and further aeration of the charge is necessary. Each aeration is attended by the formation of some acid and the consequent destruction of some lime and cyanide. After treatment, the sand shows no sign of oxidation; neither analytical nor microscopic examination can detect differences between charge and residue.

Lime, crushed in a one-stamp mill to pass a 7-mesh screen, is added to the pulp stream as it flows to the vat. Regulation is by weighing a prescribed amount of lime into the automatic stamp feeder at 2-hr. intervals. The choice of screen is determined by the rate at which the lime slakes, the intent being to exhaust the particles only when treatment is completed. Formerly small quantities of lime were added to the top of charges dur-

ing the drainage periods, to supplement that fed with the sand, but this is no longer considered to be necessary and is not done except in emergencies.

It had long been recognized that a high protective alkalinity was detrimental, but until within the past two years it had been tacitly accepted that it would be impossible to dispense entirely—or practically so—with protective alkali. Eventually this was done; the quantity of lime added was reduced to the minimum necessary to correct the acidity of the ore; extractions improved and consumption of cyanide was materially reduced.

The gratifying results are in a measure due to another change in procedure, initiated at about the same time. Reference to the treatment chart will show that solutions of two strengths are used; that stronger in cyanide is used earliest in the treatment of a charge. When these solutions appear as effluents, one portion is precipitated, the other brought to full working strength by adding cyanide and thereafter returned to the extraction of another charge of sand.

This custom is by no means unusual, yet wherever it is used the procedure seems to be to maintain the stronger solution at a fixed content of cyanide and to allow the weaker to vary in strength according as the varying losses in treatment may determine. This is wrong in principle. The critical strength is that below which the weaker solution will no longer dissolve its quota of gold, or, if dissolving it, will no longer freely yield it to precipitation. If the strength is greater than this, we may confidently expect some cyanide to be destroyed without giving compensatory service. In such circumstances, if the maximum working strength is reduced, the weaker solution will be automatically restored to the lowest economic strength; if it falls below this, a temporary increase in the maximum strength will restore it. In other words, cyanide should be added to the *strong* solution in accord with the determinations of strength made on the *weak* solutions. Thus, operating with a strong solution of variable strength and a weak solution also variable, but never far from its effective limit, it may be anticipated that little cyanide will be needlessly expended.

The following definitions are required in connection with the treatment chart (Fig. 3).

Leaching Rate is the fall of the surface of solution in inches per hour when the charge is completely covered.

Alkalinity is expressed as the number of cubic centimeters of tenth normal acid required to neutralize 100 c.c. of the solution tested.

1 unit = 0.0028 per cent. CaO.

Protective alkalinity = $(0.056a - 11.4n)$ (in terms of pounds CaO per ton) when a = alkalinity and n = per cent. NaCN.

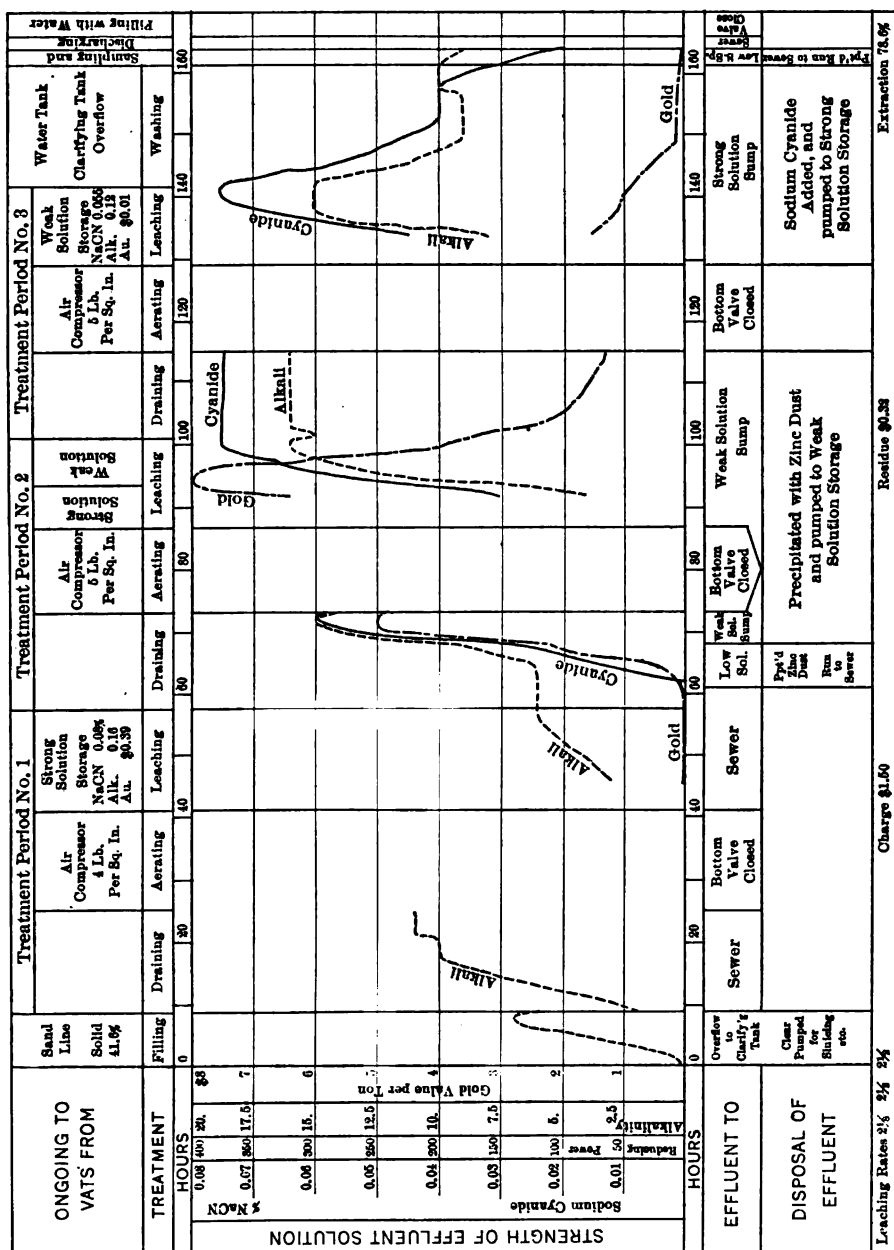


FIG. 3.—CHART SHOWING TREATMENT AT SAND PLANT No. 1. Data secured in March, 1914.

No. 1 Sand Plant: Operating Costs for 1914

Tons treated, 598,043

	Operating Labor	Other Labor	Power	Chemicals	Misc.	Total
Superintendence.....	\$0.0081					\$0.0081
Assaying.....	0.0033			\$0.0004	\$0.0006	0.0043
Transportation.....	0.0010				0.0001	0.0011
Neutralization.....	0.0034		\$0.0002	0.0093		0.0129
Classification.....	0.0086	\$0.0002	0.0007		0.0008	0.0103
Treatment.....	0.0293	0.0011	0.0054	0.0541	0.0021	0.0920
Precipitating and pump- ing solutions.	0.0055	0.0005	0.0005	0.0046	0.0006	0.0117
Heating.....	0.0007				0.0041	0.0048
Refining.....	0.0015				0.0022	0.0037
Miscellanies.....	0.0019	0.0009	0.0005		0.0024	0.0057
Repairs*		0.0121			0.0072	0.0193
Power from H. M. Co.....			0.0033			0.0033
Total.....	\$0.0632	\$0.0148	\$0.0106	\$0.0684	\$0.0202	\$0.1772

* Includes only larger items of repair work, necessitating the labor of a crew of carpenters.

Unit Consumptions per Ton of Sand Treated

Sodium cyanide.....	0.257 lb.
Zinc dust.....	0.061 lb.
Lime.....	2.48 lb.
Power.....	1.09 kw-hr.

SLIME TREATMENT

The slime plant is equipped with 30 Merrill presses, each of 90 frames, 4 ft. by 6 ft. by 4 in. One press holds 26 tons of slime, and has a capacity of about 70 tons a day.

The entire treatment is given in the presses, which successfully meet the conditions for which they were designed, holding a tight and homogeneous cake during aeration periods (which impose a most exacting requirement on the machines) and discharging readily, without requiring manual labor. In this connection it may be mentioned that all the operations incident to the filling, aerating, leaching, washing, and discharging of 30 presses—totaling about 75 charges daily—are conducted by three men on each shift. The actual time required by one of these men in the performance of every step in the treatment of a charge is slightly less than 15 min.

Treatment is in principle identical with that of the sand, periods of aeration and treatment alternating with sufficient frequency to maintain the activity of the solutions. The alkalinity is carried at a very low point, and the working solutions are of low cyanide strength, as reference to the

charts will show. Within the restrictions imposed by practice, there seems to be no solution so weak that it will not freely extract gold from the aerated slime. The problem seems to demand the passing of a definite volume of solution through the slime cake, rather than treating with solution of a definite strength or for a definite time. So long as this amount of solution is passed, the extraction is accomplished. Thus the treatment time is a function of the leaching rate.

The minimum working strength is determined by our ability to precipitate the effluent solution, and in developing this minimum much of the treatment is given with very dilute solutions, with the result that the consumption of cyanide has been reduced to an extremely low figure. For instance, a 10-day period, concluding as this is written, gives the following:

Sodium Cyanide (128 Per Cent.) Consumed per Ton of Slime Treated

In treatment.....	0.127
Added for precipitation.....	0.008
Total.....	0.135

The precipitation of the "low solution" (first and final effluents, low in both gold and cyanide) is difficult; doubly so on account of the low alkalinity; and after trial of many expedients to assist it we are not able to improve upon Carter's old system of a drip of strong solution.

This, usually added with the feed of zinc dust at the rate of 1 lb. per hour (or per 25 tons of solution precipitated) constitutes from 5 to 10 per cent. of the cyanide used. An excessive amount of zinc dust is also necessary to insure clean precipitation of these solutions, and this entails further expense, not merely in the first cost of the zinc, but in the refining of the larger bulk of low-grade precipitate. Combined, these costs consume about 25 per cent. of the profit won through the reduction of cyanide consumption, a satisfactory enough result in a plant where treatment methods have been so well established as is the case here, even though the impression may be conveyed that one branch of the work has lost efficiency.

Some assistance in this precipitation difficulty is received through the application of this barren low solution as a preliminary wash.

Treatment charts (Figs. 4 and 5) give full details of the treatment of typical fast- and slow-leaching charges. The leaching rate is stated as the number of seconds required to fill a 5-gal. bucket with effluent solution, the press being full of slime and the ongoing solution being applied at 30 lb. pressure.

Double cloths are used on the treatment presses, the lighter one being in contact with the frame. This is of unbleached muslin twill, and is in

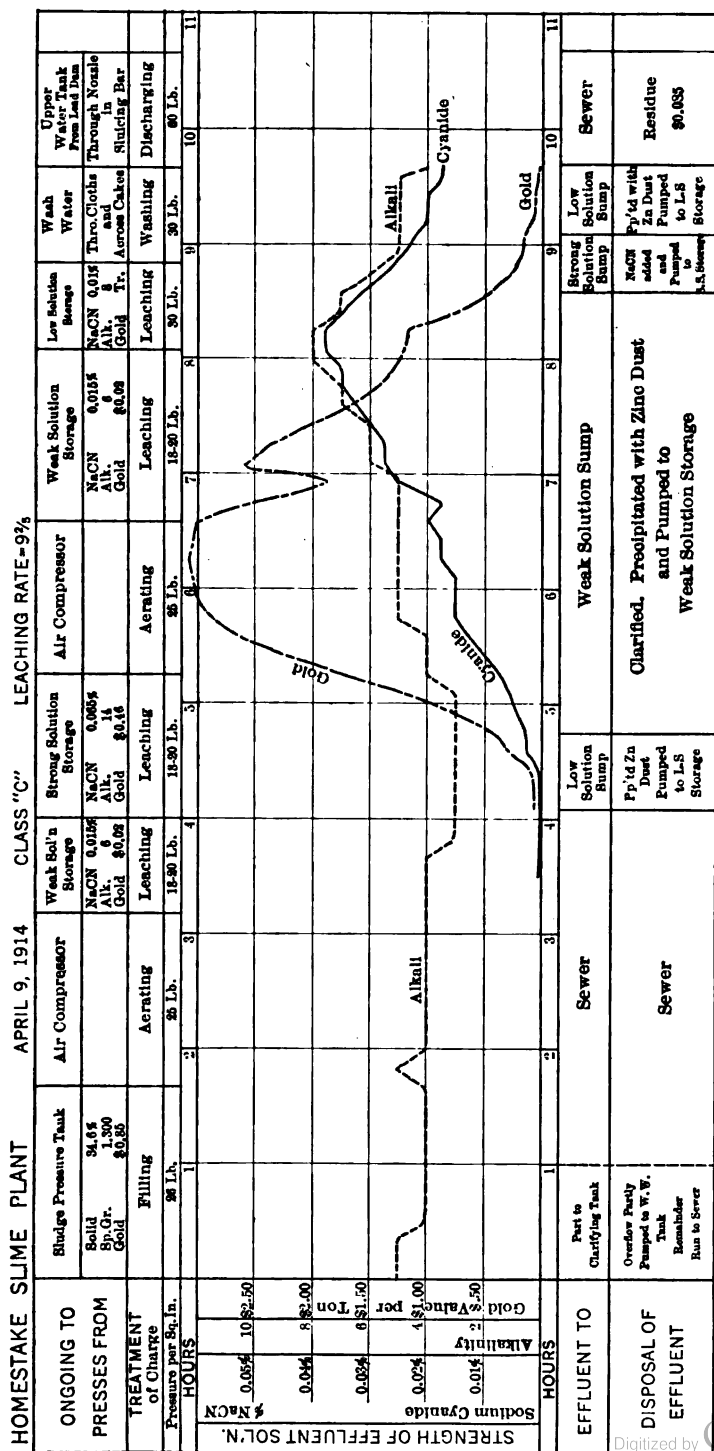


Fig. 5.—TREATMENT OF SLOW-LEACHING CHARGE, HOMESTAKE SLIME PLANT.

turn covered with a cloth of No. 10 canvas duck. Joints are made with Thermalite paint.

During the early days of the plant, when the equipment was not equal in capacity to the tonnage of available slime, a suit of cloths was used for as much as 30 months, it being preferable to lose slime rather than to stop to repair leaks. It should be remembered that the arrangement of ports and channels in the presses is such that treatment may be given in either direction, so that a single leak would in no wise affect treatment, but would merely entail the wasting from the plant of a certain amount of untreated slime. At this time, as much as 500 tons was wasted in a month. As the plant has been brought to full capacity, the cloths have been renewed, first after 18 months' service, now after 16 months; the leakage loss on the latter basis has been brought to an average of less than 40 tons monthly over an entire year, with a minimum of 8 tons in a single month. This in treating a total monthly tonnage of more than 58,000.

The "flow-gravity" system of tonnage measurements, elaborated by W. J. Sharwood (*Mining Magazine*, vol. i, No. 3, p. 226 (Nov., 1909), finds an interesting application in the estimate of slime treated at this plant. Here, even were it possible to determine the weight of the slime cakes, it would still be impossible to know the tonnage treated, since the amount of slime retained in a press after sluicing cannot be absolutely known. Certain classes of slime are more difficultly discharged than others; the stoppage of a single sluicing nozzle leaves 1.1 per cent. of the press unsluiced. Frequent testing of sluicing bars and examination of presses after sluicing protect us from loss of efficiency from this cause, so that the sluicing is never less than 96 per cent. efficient, and is at present, in my judgment, close to 99 per cent.

To at once determine the net capacity of a press, the "flow-gravity" system is used in a large way, measuring the total tonnage received at the plant during a 24-hr. period. This method is simple, thorough, and convincing, and since it is not so generally known as its value warrants, the full report of such a test is given herewith.

Procedure.—Valves on the two "sludge tanks" receiving the incoming stream of pulp are closed. The depth of slime in each tank and the time are noted. The stream of pulp is now diverted to tank A, the discharge valve from which is closed, the presses being served with pulp delivered from tank B. When tank A is filled, the time and volume are noted, the valve from B is closed, the pulp turned to tank B and the discharge valve from tank A is opened. These alternations continue throughout the period assigned for measuring. Changing, which is at intervals of about 35 min., is not instantaneous, but the actual measurements cover 90 per cent. of the total time, sufficient to insure a correct average rate.

At half-hourly intervals the stream of pulp is cut by a mechanical sampler, a 5-gal. bucket being filled and the weight of pulp determined. These determinations are averaged. A specific gravity determination is made on the average sample of slime.

Estimate of tonnage treated between noon of Apr. 8, 1915, and noon on following day;

	Feet	Cubic Feet
North Sludge Tank:		
Total depth pulp to tank.....	132.667	
Deduct, lime water.....	1.078	
Balance, slime pulp.....	131.589	
131.589 × 518.85 (cu. ft. per foot of depth)....		68,274.95
South Sludge Tank:		
Total depth pulp to tank.....	125.334	
Deduct, lime water.....	1.003	
Balance, slime pulp.....	124.331	
124.331 × 514.42.....		63,958.35
Total cubic feet for 22.05 measured hours.....		132,233.30
Estimated pulp for 24 hr.....		143,927.28
Correction for excess pulp in tanks at end of test.....		3,590

Cubic feet of pulp run to presses during test..... 140,337 +

During the test, 39 determinations were made of the weight of 5 gal. of pulp. The average of these determinations gave the following data:

Percentage of solids in pulp.....	33.68
Weight of 1 cu. ft. of pulp.....	80.80 lb.
Solids in 1 cu. ft. of pulp.....	27.146 lb.

Total solids to presses during test = $140,337 \times 27.146 = 3,809,600$ lb. = 1,904.8 tons, which was divided among 73 presses, making the average charge 26.09 tons. This scheme of tonnage measurement has been elaborated at our mills during the past eight years, and has found many useful applications. Two more of these may be cited. One, another application in a large way, was described in *Transactions of the Institution of Mining and Metallurgy*, vol. xxii, p. 201 (1912-13) from which I quote:

"At the start the sand vat was completely filled with water. A large tank (1246 cub. ft.) was arranged so that the entire overflow (equalling in volume the entering pulp) could be switched into it, and the time of filling the 1246 cub. ft. noted. This tank was filled 53 times at equal intervals during the 48 hours required to fill the sand vat, the total volume of pulp being thence calculated as 92,711 cub. ft. Meanwhile the pulp was systematically sampled by a mechanical cutter, a standardized bucket of $\frac{3}{8}$ cub. ft. capacity (5 U. S. gallons), being filled and weighed 116 times at equal intervals. The samples were all thrown upon a filter, to be dried and weighed later. The specific gravity of the dry solid was accurately determined. From these data the following computations were made:

- (b) Sp. gr. of ore 2.985. Average weight of 116 buckets [of 5 gal.] = 54.575 lb., or 81.862 lb. per cub. ft. pulp. Average sp. gr. of pulp = 1.31, whence 1 cub. ft. contains 29.062 lb. dry sand. 29.062 dry lb. sand per cub. ft. $\times$ 92,711 cub. ft. $\div$ 2000 = 1347.2 tons [sand in vat].
- (c) Total dried sand from 116 buckets ... weighed 2261 lb., or 29.24 lb. per cub. ft. pulp. 29.24 lb. per cub. ft. $\times$ 92,711 cub. ft. $\div$ 2000 = 1355.4 tons.
- (a) Tonnage for a depth of 160 in., based on former accepted weight of cub. ft. boxes = 1344.0 tons.

Maximum variation = 11.4 tons = 0.85."

Another convincing demonstration of its accuracy, under more difficult conditions, has recently been noted. A detail of stamp milling was under investigation. Tonnage measurements of the trial battery and of a standard battery were made by diverting the entire flow of a five-stamp battery into a 15-gal. container, noting the time required to fill the container and the weight of the impounded pulp. The time required to fill the container, under these conditions, is from 20 to 25 sec.; the water ratio is over 10:1. Although many repetitions carried conviction of the general accuracy of the observations, it was thought best to confirm the results by weighing all ore supplied to the batteries, this decision being based upon the double liability to a large percentage error on account of the brief sampling time and the dilute pulp. Accordingly, a scale was installed, with the result that the earlier determinations were confirmed within 2 per cent.

A statement of operating costs at the slime plant is appended. This plant was the pioneer of its type. It is interesting to note that the presses are now practically as at first designed, and that the process, in its essentials, is unchanged. Certain refinements in the press design have been dispensed with as unnecessary; a study of the operations has made possible some simplification and systematization, which have made it possible to reduce the costs each year.

Slime Plant Operating Costs for 1914

	Operating Labor	Other Labor	Power	Chemicals	Miscellaneous	Total
Superintendence.....	\$0.0093					\$0.0093
Assaying.....	0.0021			\$0.0005	\$0.0005	0.0031
Thickening.....	0.0033				0.0001	0.0034
Neutralization.....	0.0055		\$0.0004	0.0144		0.0203
Treatment.....	0.0342	\$0.0027	0.0152	0.0433	0.0119	0.1073
Precipitating and pumping solutions.....	0.0039	0.0006	0.0016	0.0096	0.0008	0.0165
Heating.....	0.0004				0.0013	0.0017
Refining.....	0.0026				0.0040	0.0066
Miscellanies.....	0.0057	0.0009	0.0012		0.0010	0.0088
Repairs*		0.0042			0.0026	0.0068
Total Operating	\$0.0670	\$0.0084	\$0.0184	\$0.0678	\$0.0222	\$0.1838

* Minor repairs are carried as Miscellanies; heavy items of repairs, necessitating the presence of carpenter's crew, are included here. Press cloths cost \$0.0100 per ton of slime treated.

Unit Consumptions per Ton of Slime Treated

Sodium cyanide.....	0.161 lb.
Zinc dust.....	0.118 lb.
Lime.....	3.84 lb.
Hydrochloric acid.....	0.393 lb., costing \$0.0093 per ton treated.
Power.....	1.15 kw-hr.

PRECIPITATION

The Merrill system of precipitation with zinc dust is used at all cyanide plants, as is the Merrill filter press of triangular section. This method has been fully discussed in the paper already cited, and the technique remains about as there described. It has been mentioned that the effect of the recent variations in treatment has been adverse to precipitation, since the tonnage of solution requiring precipitation has been increased at the same time that the content of solution in gold and cyanide has been reduced. Beyond the natural effects—increased zinc consumption and lower-grade precipitate—no difficulties have been encountered.

Double cloths are used on the frames, the outer one being taken off when the press is cleaned and replaced by the inner cloth, which is in turn replaced by a new cloth. The outer cloth is burned, the ash going to refining.

Experiments are still in progress to find the lightest and cheapest cloth that can be used for this purpose without unduly increasing the hazard of leakage.

All washing of cloths is eliminated by this system. Against the added cost of cloth may be set the advantage of security from leaks due to rotten or wrinkled cloths and the labor of washing soiled cloths.

REFINING

In its essentials, this remains as described in the earlier paper, except that after acid treatment of the lower-grade precipitate, the briquets are charged to the blast furnace instead of being fused in the cupel. The latter treatment is still the standard for precipitate of higher grade.

This change and the general scheme of treatment are shown in Fig. 6.

Acid treatment is a preliminary to furnace treatment of all precipitates. It may not be necessary, but in my judgment it is sound metallurgy, since it enhances security from stack losses and insures more fluid slags. Moreover, the decreased bulk of product to be smelted with lead decreases the time required for that operation, always more or less hazardous to the health of the workmen.

Oil is now used as a fuel in the cupel furnace, replacing wood. It is an ideal fuel for the purpose, and its use has resulted in an acceleration of nearly 50 per cent. in the rate of cupelling.

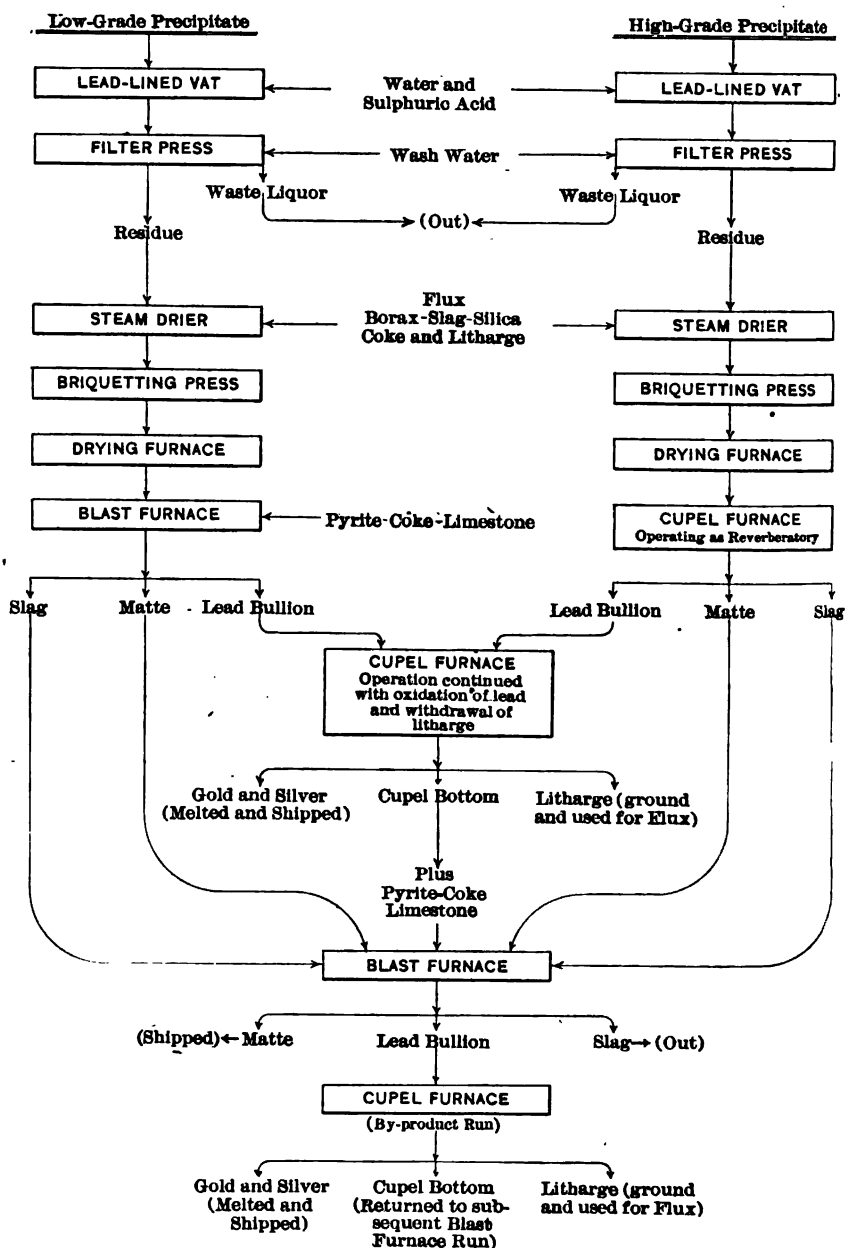


FIG. 6.—REFINING OF PRECIPITATE AT HOMESTAKE

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The British Columbia Copper Co.'s Smelter, Greenwood, B. C.

BY FREDERIC K. BRUNTON,* B.S.C., VICTOR, COLO.

I. INTRODUCTION

THE smelting plant of the British Columbia Copper Co. at Greenwood, B. C., now closed because of the decline in the price of copper due to the European war, is of special interest to metallurgists for several reasons.

It was successfully smelting in blast furnaces the lowest grade copper ore of all plants in America. In order to do so, it had to run at very high efficiency, which necessarily required a large tonnage per square foot of hearth area, together with the minimum amount of labor and other costs.

The furnaces smelted daily 2,250 tons of ore (6.62 tons per sq. ft. of hearth area), carrying 0.85 per cent. of copper, at a smelting cost of \$1.18 per ton. The entire plant required 130 men to operate it and keep up repairs, showing a labor efficiency of about 17.5 tons per man per day.

In the present paper, the method of obtaining these results is shown. Most of the information contained in this description I obtained as Assistant Superintendent, during the two years preceding the present shutdown. Other sources of information to which I am indebted are: The British Columbia Copper Co., Ltd., by Alfred W. G. Wilson, in *The Copper Smelting Industry of Canada*; Greenwood Smelting Works, by J. E. McAllister, *Engineering and Mining Journal*, vol. xci, pp. 1011 to 1015 (May 20, 1911); and Description of the Copper Smelter of The British Columbia Copper Co., by W. L. Bell, in *Transactions of the Canadian Mining Institute*, vol. xvi, pp. 151 to 154 (1913).

II. LOCATION AND CONNECTIONS

The smelter of this company is about half a mile south of Greenwood, in the Boundary district of British Columbia, on the Columbia & Western branch of the Canadian Pacific Railway. It was originally built in 1900 by Paul Johnson, to handle 600 tons of ore a day in two small furnaces of 300 tons capacity; but in 1907 the old furnaces were torn out, the plant was remodeled, and three blast furnaces, 48 by 240 in. at

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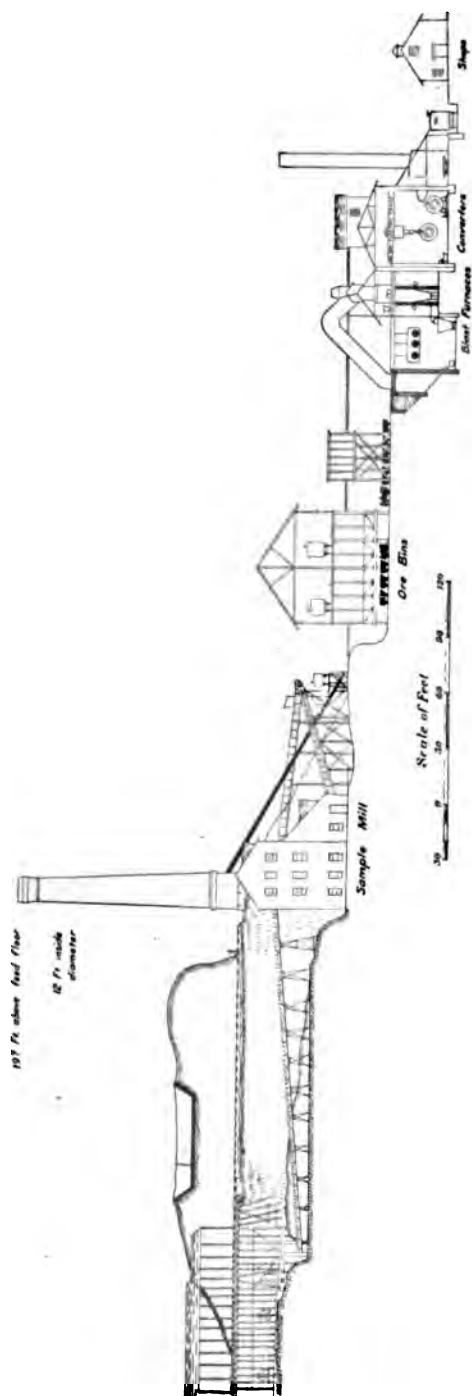


FIG. 2.—SECTION THROUGH REDUCTION WORKS AT GREENWOOD.

the tuyères, were installed. In 1910, two of these were enlarged to 51 by 360 in. and the other to 51 by 240 in., making the total smelting capacity about 2,400 tons per day.

The Canadian Pacific Railroad delivered ore and coke to the smelter, which is situated on a hillside above the valley, by eight of the nine spurs of track which enter the plant at three different levels; the ninth spur, coming into the plant on the lowest level, took care of the incoming supplies, converter slag, and copper shipments. The ore as received

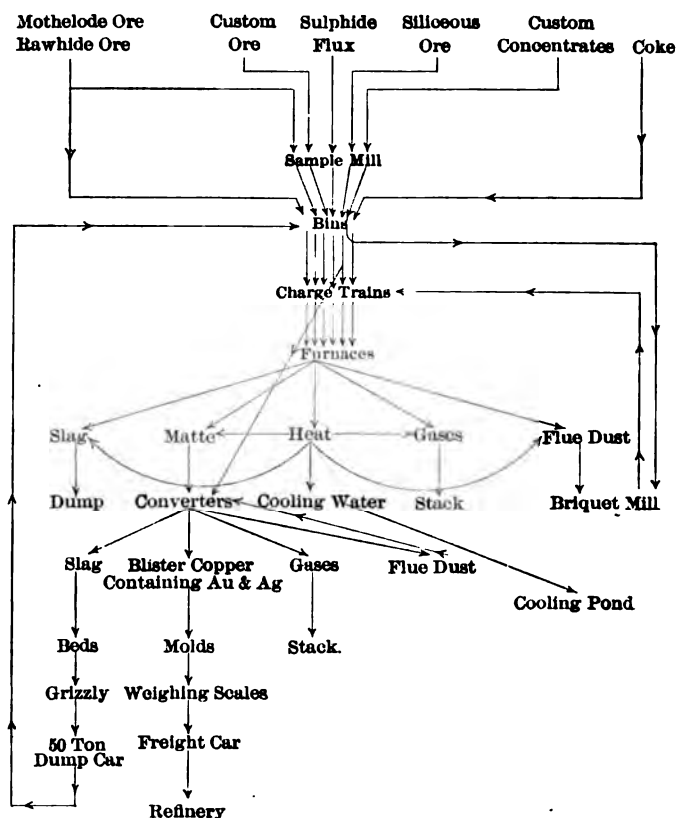


FIG. 3.—GENERAL FLOW SHEET OF THE BRITISH COLUMBIA COPPER CO.'S SMELTER, GREENWOOD, B. C.

was either dumped into the bins above the sampling works or directly into stock bins above the charging floor of the smelter. The sampling mill was served by three of the nine spurs, one of which, running over a 200-ton bin, brought ores from the company's own mines, while another, running over two small bins of about 60 tons capacity, was used for custom ores.

On the intermediate level, 48 ft. below the tracks serving the sampling

mill, are five spurs, four of which run over the main stock bins and one under the discharge belt of the sampling mill, so that rejects from the mill can be switched to the bins very handily.

The main spur coming into the plant at this level is equipped with a standard 100-ton self-registering track scale, so that all material can be weighed if desired.

Fig. 1 is a plan and Fig. 2 a section of the plant. The general flow sheet of the smelter is shown in Fig. 3 and the flow sheet of the sampling mill in Fig. 4.

III. SAMPLING, STORAGE, AND CHARGING

1. *Sampling Mill*

The building is 65 by 79 ft. in area, and three stories high. The ore from the sample bins, whether custom or company ore, was discharged through chutes into a 36 by 24 in. Farrell-Bacon crusher, crushing to 8 in., driven from a countershaft by a 100-hp. Allis-Chalmers Bullock, variable speed, 2,200-volt, induction motor. The crushed ore went to the sample mill on a 30-in. belt conveyor, 225 ft. between centers, driven from the mill line shaft. If it was company ore, this conveyor delivered it into a bucket sampler which cut out 11.5 per cent. for a sample, while the reject, falling on a 26-in. belt conveyor, 116 ft. between centers, driven by a 15-hp., 550-volt, motor, was delivered to a standard 50-ton steel dump car which, when full, was switched to the smelter storage bins. If the ore from the jaw crusher was custom ore, it passed by the bucket sampler and was delivered through a chute into a No. 5 Gates crusher, crushing to 4 in. The 11½ per cent. sample of company ore was delivered to the same crusher by the bucket sampler; and either ore, after leaving the crusher, went through a double-spouted Johnson 48-in. sampler, taking a 10 or a 20 per cent. cut, as desired, the reject going to a bin which discharged on to a 24-in. belt conveyor 120 ft. long, driven by a 10-hp., 550-volt, induction motor, delivering to the dump car mentioned above.

The sample from the 48-in. Johnson sampler was passed through a No. 2 Gates crusher, crushing to 2 in., and then a 40-in. Johnson sampler taking a 20 per cent. cut. This and all other rejects were delivered to the 24-in. belt conveyor above referred to, while the sample was elevated by a bucket elevator to a No. 0 Gates crusher, crushing to 1 in., and then passed through a set of 10 by 18 in. rolls, crushing to 0.5 in., after which it was delivered to a 24-in. three-spouted Johnson sampler taking a 20 per cent. sample. The reject went to the dump car and the sample to a sample car in the sample room, from which it was taken to a steel floor and cut down with a Case riffle to about 30 lb., which was then taken to the bucking room, where the sample was alternately crushed and riffled until the pulp was of suitable size for the assay office.

2. Bins

The 32 bins are built in two lines of 16 each, parallel to and touching each other, and served by four parallel standard-gauge spurs, these being four of the five entering the smeltery on the intermediate level. The back or south end of this line of bins consists of 15 bins of 600 tons capacity each, 6 of 300 tons, and 4 of 100 tons, used for ore, while at the front or north end are 10 bins for coke, each of which will hold 180 tons, and one large box bin for clay, making the total capacity of the bins 11,200 tons of ore and 1,800 tons of coke. Coke for stock purposes was dumped on the ground from the fifth railway spur, which also served the sampling mill, and about 6,000 tons of coke could be stored there in an emergency. The ore and coke were all dumped by a crew of four men and a foreman, who also looked after the sampling-mill crew.

3. Charging Equipment

The ore and coke were drawn from the bins into charging cars running on four parallel, 36-in. gauge, electric tramway lines on the feed-floor level, 31 ft. above the top of the ore bins. The bins and furnaces being on opposite sides of an oval, the tracks are so arranged that, with switches, a train may run on any track under the bins and come in to feed on either side of any furnace, the trains running completely around the oval. There was a charging train for each furnace, consisting of four ore and four coke cars for each of the 30-ft. furnaces and three ore and three coke cars for the 20-ft. furnace, each train being, roughly, twice the length of the furnace it fed. Each track is equipped with three sets of automatic platform scales, for weighing the ore and coke, placed on a tangent under the bins.

The cars, approximately V-shaped in cross-section, are side-dumping, being released by a foot trip in the bottom. They have a capacity of 50 cu. ft. of material and, when made up into a train, were hauled by a 7.5-ton electric locomotive, handled by a train crew of three men, a motorman, a head loader, and a second loader.

4. Charging

The crew usually made up the charge by loading the coke in the last half of the train and weighing it; then running ahead and drawing the ore into the front cars and weighing it on the nearest platform scale; and then running around the oval to the furnaces, where, with the help of the feeder, the coke cars were dumped first; the train was then backed up and dumped the ore, after which it proceeded around to the bins for another charge. The charge usually consisted of 5,000 lb. of ore per car and about 13 per cent., or 650 lb., of coke. The loaders were very

expert at estimating their charge as they drew it from the bins and usually ran in very nearly the amount required, so that when it was weighed they only had to take off or add from 5 to 10 lb.

IV. BLAST-FURNACE DEPARTMENT

1. Blast Furnaces

The blast-furnace building is 150 ft. long by 60 ft. wide and contains three water-jacketed blast furnaces placed end to end, with space between them for the minor axis of a 10 by 18 ft. oval settler. The two outside furnaces, Nos. 1 and 3, are each 51 by 360 in., while the middle one, or No. 2, is 51 by 240 in. in area at the tuyères. The vertical distance from the center of tuyères to the feed floor is 16 ft., and to the sole plate 37 in., the other furnace dimensions being as follows:

	30-ft. Furnace	20-ft. Furnace
Hearth area.....	127.5 sq. ft.	85 sq. ft.
Center tuyères to tapping floor.....	5 ft. 3 in.	5 ft. 3 in.
Height bottom jackets.....	9 ft. 0 in.	9 ft. 3 in.
Width of side jackets.....	3 ft. 4 in.	3 ft. 4 in.
Width of end jackets, bottom.....	3 ft. 8 in.	3 ft. 8 in.
Width of end jackets, bottom.....	6 ft. 2 in.	6 ft. 2 in.
Number of tuyères.....	72	48
Diameter of tuyères.....	4 in. bushd to $3\frac{1}{4}$ in.	4 in.
Area of tuyères.....	597.4 sq. in.	602.9 sq. in.
Tuyère area per sq. ft. of hearth area.....	4.65 sq. in.	7.09 sq. in.
Center line to center line tuyères.....	9.25 in.	9.25 in.
Water space in jacket, 4 in.; plate used on inside $\frac{5}{8}$ in.; on outside, $\frac{3}{8}$ in.		

The furnaces and settlers are set upon concrete foundations about 4 ft. above the converter floor. The settlers are 10 ft. wide, 18 ft. long, and 4 ft. 8 in. deep, lined on the bottom with a layer 6 in. deep of chrome bats ground up and tamped in with clay, and this covered with a course of firebrick set on end, and on the sides with two courses of chrome brick, backed by crushed silica, for about 22 in. above the bottom. Above that line silica or firebrick are used instead of the chrome brick, except directly under the slag and matte spout of the furnace, and under the settler slag-discharge spout. The top of each furnace is connected with a dust chamber of 550 sq. ft. cross-section by a sheet-steel downtake, 7 ft. in diameter, in the form of an inverted V. This dust chamber is hopper bottomed, and the dust was drawn off in a small steel mine car and trammed to the briquet mill, where it was mixed with about 3 per cent. water and made into briquets in a White three-plunger press, having a capacity of 5 tons per hour, and driven by a 40-hp., 550-volt, induction motor. The briquets were fed back to the furnaces again as extra coke when needed, since they contained about 70 per cent. of coke dust. The flue dust recovered amounted to

about $1\frac{1}{2}$ per cent. of the total charge fed to the furnaces. The gases from the dust chamber passed through an inclined flue 180 sq. ft. in cross-section to the stack, which is 121 ft. high, with an inside diameter of 12 ft. The total height of stack above the top of the furnace-charge floor is 200 ft., which, under normal conditions, gave a good draft and kept the feed floor free from fumes.

2. Operation of the Furnaces

The ore and coke were charged into the furnaces from the side the furnace door being lifted by compressed air. The cars of coke were dumped first, and the train was then backed up and the ore dumped. The charge fell against baffle plates, suspended on both sides of the furnace on 4-in. heavy hydraulic pipe which is supported by the end castings of the furnace on a level with the charge floor. To insure further the even distribution of the feed, the furnaces were fed first on one side, then on the other. When the plant was running under normal conditions, the two 30-ft. furnaces smelted from 12 to 16 5,000-lb. charges an hour, or about 112 charges per shift, each, and the 20-footer from 8 to 12 5,000-lb. charges per hour, or 80 per 8-hr. shift, making a total of 912 charges of 5,000 lb., or 2,250 tons, per day. The blast varied from 16 to 24 oz., according to conditions, and was supplied by three No. 10 Roots blowers, each having a capacity of 25,000 cu. ft. per minute running at 88 rev. per minute and belted by a 40-in. leather belt to a 300-hp., 2,200-volt, induction motor. The blowers are so arranged that, through a common header, any blower could be used on any furnace, the air going in iron pipes, 43 in. in diameter, from the header to the furnace bustle pipe, which extended around both sides of the furnace.

The slag and matte ran out of the furnaces at the end through a water-jacketed, copper breast jacket and thence through a water-jacketed spout with a water-cooled copper lip, into the settler. The spouts are set to have about a 7-in. trap. The slag overflowed continuously into a cast-iron pot of 250 cu. ft. capacity, holding from 20 to 25 tons of slag, and mounted on standard-gauge trucks, in which it was hauled to the dump by a 35-ton, 240 hp., Baldwin-Westinghouse locomotive, and there dumped by means of a 15-hp., D. C. motor operated from the cab of the locomotive through "jumpers." While one pot was being removed from the settler and another one spotted, the slag was caught in a cast-iron spoon, supported upon a swinging arm and operated by hand from the side of each settler, which intercepted and received the slag stream and dumped its contents into the empty pot after the change was completed.

3. Furnace Charge

An average furnace charge was about as follows:

Ore	Lb.	SiO ₂		Fe		CaO		S		Cu	
		Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.
Motherlode.....	2,500	32	800	18	450	22	550	2	50	1.0	25.0
Rawhide.....	1,600	36	576	16	256	19	304	3	48	1.2	19.2
Napoleon.....	500	30	150	30	150	7	35	17	85	0.2	0.5
Republic, SiO ₂	400	80	320			7	28				
Total.....	5,000		1,846		856		917		183		44.7

This gave approximately 4,300 lb. of slag carrying off 0.23 per cent. or about 10 lb. Cu, and leaving 34.7 lb. for the matte, which required one-fourth of its weight, or 8.7 lb., of sulphur. By reason of the poor construction of the furnaces for handling ores so low in sulphur, 88 per cent., or 160 lb., of the sulphur was volatilized in the furnace, leaving 23 lb. for the matte. Since the Cu took 8.7 lb. of S, there was left for iron 14.3 lb. of S, requiring 1.75 times its weight, or 25.0 of iron. The matte, therefore, contained: Cu, 34.7; S, 23.0; and Fe, 25.0 lb., and assayed 42 per cent. Cu. The slag ran approximately as follows: SiO₂, 42.7; Fe, 19.7; CaO, 21.2; Al₂O₃, 9.0; and Cu, 0.23 per cent.

In the above charge the most variable factor was the Motherlode ore, in which SiO₂ varied from 17 to 38, iron from 15 to 33, CaO from 7 to 22 per cent. Since this ore would change, sometimes in an hour, from a lime gangue to an iron gangue carrying about 30 per cent. magnetite, the furnaces required constant watching. The storage facilities were so small that the ore was always smelted before the assays were out; and this made it necessary to be able to determine the composition of the charge by the physical appearance of the matte and slag and the condition of the furnaces so that the charge could be changed to meet requirements at a moment's notice.

The labor required to operate the three furnaces per shift was as follows:

Kind of Labor	Number of Men	Wages per Shift	Total Wages per Shift
Shift bosses.....	1	\$5.25	\$5.25
Furnace men.....	3	4.00	12.00
Furnace helpers.....	3	3.00	9.00
Slag motorman.....	1	3.40	3.40
Slag switchman.....	1	3.00	3.00
Charge motormen.....	3	3.15	9.45
Head loaders.....	3	3.15	9.45
Second loaders.....	3	3.00	9.00
Feeders.....	1	4.00	4.00
Binman.....	1	2.75	2.75
Power house.....	1	3.40	3.40
Total.....	21		\$70.70

In addition, there were, on the day shift only, four men in the briquet mill and four men grading on the slag track. Of these, six received \$2.75 and the other two \$3.50 a day, making a total of \$23.50. These two crews were also used, whenever needed, in general roustabout work around the plant; so that not all of their time was really chargeable to the furnaces.

4. *Résumé of Furnace Operating Data*

Tons smelted per day.....	2,250.
Tons smelted per square foot of hearth area, average...	6.62
Tons smelted per square foot of hearth area, maximum.	8.70
Tons smelted per man per day.....	35.70
Cu on charge, per cent.....	0.8 to 1.2
Cu in matte, per cent.....	30.0 to 45.0
Cu in slag, per cent.....	0.22 to 0.27
S on charge, per cent.....	2.00
S burnt off, per cent.....	85.00 to 90.00
Coke used on charge, per cent.....	12.00 to 14.00
Coke ash, per cent.....	20.00 to 28.00
Blast, cubic feet per minute.....	25,000
Blast, temperature.....	Atmospheric
Cooling water for jackets, gallons per minute...	2,500
Men per 8-hr. shift.....	21
Matte, per cent. of total charge.....	1.65
Matte, specific gravity.....	5.0
Slag, per cent. SiO_2	38 to 45
Slag, per cent. Fe.....	13 to 20
Slag, per cent. CaO	20 to 26
Slag, per cent. Al_2O_3	6 to 9
Slag, specific gravity.....	3.2

V. CONVERTER DEPARTMENT

The converter building is 150 ft. long by 44 ft. in width, stands parallel to and about 4 ft. lower than the blast-furnace tapping floor, and contains the following equipment:

Two hydraulic converter stands; seven converter shells, each 84 by 126 in.; a 40-ton Niles electric traveling crane, a 6-ft. Carlin silica mill driven by a 40-hp., 550-volt, induction motor, a pneumatic tamping machine, copper casting trucks, slag and matte pots, and a set of platform scales for weighing the blister copper. The converters were lined with siliceous gold ores from Republic, Wash., or from the Snowstorm mine in Idaho, mixed with local clay, as a binder, in the Carlin mill. The life of a lining was from two to three charges.

The matte, containing from 30 to 45 per cent. Cu, was tapped from the settlers into 5-ton ladles, handled by the Niles crane, and poured into one of the converters, where from two to three ladlefuls were blown to blister copper at one heat. The slag was poured off into old matte

ladles and was cast in hollows in the ground, into buttons, which when cool enough, were lifted by the crane, placed on grizzlies over 50-ton steel dump cars, and there broken up, with hammers, small enough to pass through the grizzlies. These cars, when full, were switched out over the ninth and lowest railway spur, which is 61 ft. below the intermediate level referred to before, and taken up to the main smelter bins, where the material was dumped and subsequently smelted in the furnaces.

The blister copper was cast into slabs, weighing approximately 360 lb. each, in cast-iron molds mounted on trucks (five molds to each truck) which ran on tracks under the converters. The copper slabs were weighed; loaded in box cars spotted on the lower spur, near the copper scales; and shipped East to be refined. This lower spur runs between the converter building and the machine shops and warehouse, and was also used for receiving incoming supplies.

The gases from the converters passed off into an iron flue, connected with an expansion chamber of 281 ft. cross-section, and thence into a steel stack 72 in. in diameter and 75 ft. high; the flue dust collected being fed back into the molten converter charge.

The converter department produced per day about 30,000 lb. of blister copper, carrying about 7 oz. of gold and 30 of silver per ton. It required a crew of 21 men which, divided as follows into two 8-hr. shifts, was able to handle all the matte produced:

Kind of Labor	Day Shift,	Afternoon Shift
	7 A.M. to 3 P.M.	3 P.M. to 11 P.M.
Foremen.....	1	0
Converters.....	2	2
Crane.....	2	2
Laborers.....	3	1
Lining.....	5	3
Total.....	13	8

VI. POWER

Power was purchased from the West Kootenay Light & Power Co. on a wattmeter basis and usually came from the Bonnington Falls power plant, about 85 miles away. It was brought by two 3-phase, 60-cycle, 60,000-volt high-tension lines, to their Greenwood substation, where it was transformed to 2,200 volts, at which intensity it was delivered to the smelter power house, to be used direct, or further transformed to suit the individual motor supplied.

The power house is divided into two parts, one of which contains the Roots blowers for the furnaces, already described, and the other a cross-compound Nordberg blowing engine for the converters, having a 40-in. air cylinder, 42-in. stroke, and a capacity of 5,000 cu. ft. per minute at from 8 to 10 lb. pressure per square inch, driven with a rope drive by a 300-hp., 2,200-volt, Canadian General Electric variable speed

induction motor. A compound duplex Canadian Rand compressor of 340 cu. ft. capacity, belt-driven by a 50-hp., 550-volt, induction motor running at 150 rev. per minute, supplied air at 80 lb. pressure for the converter tampers, the blast-furnace doors, the pneumatic hammer in the blacksmith shop, and for odd jobs around the plant where compressed air was required. Direct current at 250 volts for the crane, slag locomotive, charge-train motors, and an electric arc for burning frozen tap holes and furnace connections, was supplied by two of three motor-generator sets working in parallel, one of 75 kw., and the other of 150 kw. capacity, the third, of 100 kw. capacity, being held in reserve. The Gould hydraulic accumulator, with a ram 24 in. in diameter and a 10-ft. stroke, for tilting the converters, was operated by a triplex plunger pump, 5 in. in diameter, with an 8-in. stroke, running at 60 rev. per minute, with a capacity of 120 gal. per minute at 200 lb. pressure, belt-driven by a 20-hp., 550-volt, induction motor.

VII. GENERAL MECHANICAL EQUIPMENT

The plant has a well-equipped machine and blacksmith shop, where almost all repairs could be made. The shop contains one large and one small Bertram lathe, one large and one small radial-arm drill press, one large and one small pipe threader, and a planer, a bolt cutter, splitting shears, and a $1\frac{1}{2}$ -ton air hammer, together with forges, and all ordinary tools of an up-to-date shop, including a portable oxyacetylene welding outfit. There is also a carpenter shop and an electrical shop, where motors are rewound and general repairs made.

The mechanical crew consisted of a master mechanic, one machinist, one blacksmith, one carpenter, two electricians, one blacksmith helper, one carpenter helper, and seven machinist helpers, making a total of 15 men to keep the entire plant in repair.

VIII. WATER SYSTEM

A concrete reservoir 144 ft. above the converter floor, holding 150,000 gal. in reserve in case of fire or break-outs around the furnaces, is kept filled by a two-stage turbine pump directly connected to a 100-hp., 550-volt, induction motor, located on Boundary Creek.

Water from Copper Creek is caught in a pond, from which it is pumped through a 14-in. steel main to a set of three tanks of 220,000 gal. total capacity, located above the furnaces, by two of three direct-connected Byron-Jackson centrifugal pumps operated by one 50-hp., 550-volt, induction motor, and two 40-hp., 550-volt, induction motors, respectively. The water from these tanks was used for jacket cooling, which required about 2,500 gal. per minute, and was then returned to

the pond, where, under normal conditions, it cooled off enough to be used over again, but in very warm weather or in winter when there was practically no cold water entering the pond from Copper Creek it was necessary to cool the water in the tanks by the addition of water pumped from Boundary Creek by a single-stage turbine pump directly connected to a 50-hp., 550-volt, induction motor. The piping of the water system is arranged as far as possible in independent circuits, to allow the shutting off of any section for repairs.

IX. ASSAY OFFICE

A fully equipped assay office, for handling the ores received at the smelter, as well as the slags, mattes, and blister copper, is situated in the general office building, a short distance to the east of the smelter proper.

X. APPENDIX

The following tables and the drawings will give a better idea of the foregoing description:

**A. BRITISH COLUMBIA COPPER CO. SMELTER. AVERAGE DAILY REPORT
FOR MAY, 1913. TWO FURNACES, NOS. 1 AND 3**

Labor

FURNACES	Foremen	Furnaces	Charge track	Slag Track	Briq. Mill	Power House	Totals
Shift 1.....	1	4	8	2	4	1	20
Shift 2.....	1	4	8	2	..	1	16
Shift 3.....	1	4	8	2	..	1	16

CONVERTERS	Foremen	Converters	Crane	Labor	Lining	
Shift 1.....	1	4	2	3	5	15

SAMPLING	Foremen	Sample Mill	Bins	Bucker	
Shift 1.....	1	2	5	1	9

SURFACE	Warehouse	Slag Track	Office Boy	Chauffeur	Janitor	Nipper	
Shift 1.....	2	3	1	1	1	1	9

MECHANICAL DEPARTMENT	Foremen	Blacksmith Shop	Machinist	Machinist Helpers	Carpenters	Electricians	
Day Shift.....	1	2	1	9	1	2	16

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Smelter Expenditures

	Quantity	Furnaces	Converters
Clay.....	\$6.45	\$0.60	\$5.85
Wages.....	326.90	269.20	57.70
Supplies.....	85.35	82.05	3.30
Wood ^a	13.15	4.50	8.65
Coke.....	1,257.85	1,257.85	
Power.....	77.50	63.95	13.55
B. S. coal.....	0.70	0.50	0.20
Flux.....	164.30	164.30	
Total.....	\$1,932.20	\$1,842.95	\$89.25

Ore treated, tons.....	1,695.16
Blister Cu, pounds.....	22,009.40
Cost per ton ore to produce matte.....	\$1.084
Cost per pound Cu for converting.....	0.00391
Coke used, tons.....	206.5
Coke used, per cent.....	12.18
542 tons per man per month.	

Supplies.....	\$85.35	Foremen.....	6
Wages.....	326.90	Men.....	95

 Totals.....\$412.25

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^a NOTE.—The cost of wood for the blast furnaces has since been reduced to \$1 a day by the use, on the furnace spouts, of iron covers, lined with fireclay.

B. SMELTER OPERATIONS. AVERAGE DAILY REPORT FOR A MONTH WHEN THREE FURNACES WERE RUNNING

Delays: No. 3 was down 4 hr. on the 12th for repairs, and 8 hr. on the 21st, short of ore.

Labor

FURNACES	Foremen	Furnaces	Charge Track	Slag Track	Briquet Mill	Power House	Totals
Shift 1.....	1	6	11	3	4	1	26
Shift 2.....	1	6	11	3	..	1	22
Shift 3.....	1	6	11	3	..	1	22

CONVERTERS	Foremen	Converters	Crane	Labor	Lining	
Shift 1.....	1	2	2	3	5	13
Shift 2.....	..	2	2	1	3	8

SURFACE	Warehouse	Slag Track	Office Boy	Chaufeur	Janitor	Nipper	
Shift 1.....	2	4	1	1	1	1	10

SAMPLING	Foremen	Sample Mill	Bins	Buckers	
Shift 1.....	1	3	6	2	12

MECHANICAL	Foremen	Blacksmith Shop	Machinist	Helpers	Carpenters	Electricians	
	1	2	1	9	2	2	17

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Costs

	Furnaces	Converters	Total
Wages.....	\$333.63	\$73.53	\$407.16
Supplies.....	66.72	18.38	85.10
Wood.....	1.50	13.60	15.10
Clay.....	0.55	5.33	5.88
Coke.....	1,892.79		1,892.79
Power.....	73.40	14.97	88.37
B. S. coal.....	0.63	0.13	0.76
Flux.....	253.56		253.56

Total.....	\$2,622.78	\$125.94	\$2,748.72
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Ore treated, tons.....	2,224.2
Blister Cu, pounds.....	26,260.0
Coke used, tons.....	315.8
Coke used, per cent.....	13.70

Foremen.....	5	Average men per day.....	125
Men.....	125	Tons smelted per man per day.....	17.8
	130	Tons smelted per man per month..	551.6

C. TOTAL COSTS

The following costs do not include overhead expenses, depreciation, or insurance

Cost per ton of smelting ore to matte*	\$1.18
Cost per pound of copper of converting matte to blister.	0.0048
Cost per ton of copper of converting matte to blister.	9.60
Cost per ton of smelting ore to blister copper.	1.23
Cost per ton of copper to produce blister copper.	0.105
Cost of coke per ton of ore smelted to matte.	\$0.851
Cost of flux per ton of ore smelted to matte.	0.114
Cost of labor per ton of ore smelted to matte.	0.15
Cost of power per ton of ore smelted to matte.	0.033
Cost of supplies per ton of ore smelted to matte.	0.03
	\$1.178

Cost of coke per ton f.o.b. smelter bins.	\$6.00
Cost of flux per ton f.o.b. smelter bins.	2.75
Cost of power per kilowatt-hour.	0.0065

Distribution of Smelter Payroll for Same Month, and Cost of Labor per Ton of Ore Smelted

	Payroll Distribution	Cost of Labor per Ton of Ore Smelted
Sample mill.	\$318.05	\$0.00462
Bins.	729.35	0.01060
Briquetting.	376.65	0.00546
Furnaces.	6,508.35	0.0958
Slag disposal.	1,413.65	0.0206
Linings.	615.60	0.0078
Converters.	1,016.85	0.0147
Crane.	277.25	0.00403
Water system.	224.65	0.00326
General surface.	430.15	0.00624
Power house.	585.60	0.00850
Total.	\$12,496.15	\$0.18161

Briquet mill handled 1,057 cars of blast-furnace flue dust and made 398 tons of briquets.

Briquets cost \$0.945 per ton for labor.

* Note.—The furnaces were slowed up with an excess of silica on the charge because of shortage of ore, hence the higher cost per ton of ore smelted to matte. They only smelted 6.55 tons per square foot of hearth area against 6.66 tons per square foot when the cost of smelting was \$1.084.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Possible Occurrence of Oil and Gas Fields in Washington

BY CHARLES E. WEAVER,* PH. D., SEATTLE, WASH.

(San Francisco Meeting, September, 1915)

DURING the past few years there has been considerable activity in prospecting for oil and gas in several parts of western Washington. From time to time seepages of oil or emanations of gas have been reported from various places. Several companies have been organized for the purpose of drilling, but only a part of these have actually begun operations. In certain localities very small seepages of oil occur, but up to the present time no oil or gas in commercial quantities has been obtained.

In order to determine the conditions which may be favorable or unfavorable to the occurrence of petroleum it becomes necessary to know the character and extent of the geological formations in which such deposits might occur. In other regions where producing wells have been developed it is possible to know what formations contain the oil and what structural conditions influence its concentration in reservoirs or zones. It is usually possible to determine those strata which contained the materials from which the petroleum may have been derived and also the position and distribution of the more porous sandstone belts into which it may have collected. Detailed stratigraphic surveys may even determine the depth at which a certain known oil-bearing zone may be tapped at any particular place. It is also possible to recognize those formations which are presumably barren of oil, as well as those which are absolutely so.

In new regions far distant from any of those which are now producing or have produced in the past, it becomes absolutely impossible to foretell whether oil may exist in commercial quantities or not. Assuming that such deposits do exist, it is extremely difficult to predict at what depth the producing zone or zones occur. However, in prospecting in a new and unexplored region, use should be made of such laws as have been found to govern the occurrence of petroleum deposits in other well-developed fields. Such deposits in their world-wide distribution are found almost exclusively associated with sedimentary rocks. Areas composed predominately of igneous and metamorphosed rocks are to

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be regarded as extremely unfavorable localities for the occurrence of commercial deposits of oil or gas. Assuming that such oil deposits are present in an undeveloped district, they would not necessarily be found associated with all sedimentary formations present. In order that oil or gas may occur it is necessary that former organic material be present, or at one time have been present, which could have acted as a source for such deposits. Conditions favorable for the imbedding of such organic material and the requisite action of bacteria should have existed. After a formation has been developed which could act as a source for petroleum deposits there should also exist porous belts of strata, such as sandstones, which could act as a retaining zone of saturated sands. In order to prevent the escape of deposits from such a zone, there should be some relatively impervious capping. Assuming that all the foregoing conditions have been fulfilled, such strata should have been subjected to structural movements so as to develop anticlinal and synclinal folds or a doming, in order that the specific gravity relations between the oil deposits and underground waters would allow the former to segregate beneath the higher portions of the anticlines or domes. It would further be necessary that erosion should not have proceeded far enough to cut through the impervious capping and allow the various constituents of the petroleum deposits slowly to volatilize. In discussing the possibilities for the occurrence of petroleum deposits in Washington it is necessary to inquire to what extent the above mentioned conditions obtain.

Topographically the State of Washington is usually considered as being divided into six provinces. These are: The great basaltic plateau or basin area; the Okanogan Highlands; the Cascade Mountains; the Puget Sound Basin; the Olympic Peninsula; and the Coast Ranges of southwestern Washington. The Great Basin area consists of Tertiary formations which for the most are composed of lava flows of enormous thickness and interbedded sands and clays of fluvial and lacustrine origin. It is absolutely certain that no marine sediments were deposited in this region during the Tertiary. These deposits rest upon an extensively eroded floor consisting of pre-Tertiary igneous and metamorphic formations. The Tertiary rocks have been thrown into shallow synclines and anticlines. The Tertiary lava flows may be entirely eliminated as a possible source for commercial quantities of oil or gas. The interbedded sediments contain no marine organic remains. The only fossil remains which have been seen are those of leaves and carbonized fragments of wood. It is probable, however, that a few species of lacustrine animal life existed in the Tertiary lakes. No organic materials which could have acted as a source for petroleum deposits appear to have been associated with the Tertiary formations of the Great Basin area. Because of these facts this area should be re-

garded as very unfavorable for the finding of commercial deposits of petroleum.

The Okanogan Highlands embrace the northeastern portions of the State and are composed of Paleozoic, Mesozoic, and Tertiary formations. The rocks of the Paleozoic and Mesozoic consist of sandstones and shales together with intercalated lava flows, all of which have been extensively metamorphosed into quartzites, slates, schists, and greenstones. The entire series has been invaded by great batholithic masses of granite and granodiorite. In places resting unconformably upon these are lavas and lacustrine deposits of Tertiary origin. As in the case of the lavas of the Great Basin region, all of the igneous rocks in the Okanogan Highlands may be eliminated as a possible source for deposits of petroleum. The older sedimentary rocks of the Paleozoic and Mesozoic may possibly at one time have carried oil, although there is no evidence suggesting that such is the case. However, if they did, such deposits would have been destroyed by subsequent metamorphism. The Tertiary sediments are not metamorphosed but are of non-marine origin. Fossil plant remains occur, but no other organic remains are abundant which could have served as a source for commercial deposits of petroleum. From such evidence as is available concerning the geology of the Okanogan Highlands there is nothing to warrant the supposition that commercial deposits of petroleum may be present. The facts point strongly against its occurrence.

The Cascade Range extends almost due south from the British Columbia line into Oregon. The main part of the range attains elevations of from 4,000 to 6,000 ft. Resting upon the dissected surface of this mountain mass are later volcanic cones such as Mount Ranier, Baker, and Adams. The northern portion of the range is composed, as in the case of the Okanogan Highlands, of a great series of metamorphic formations of Paleozoic and Mesozoic age. Lying upon these older rocks or folded down into them are residual patches of Tertiary lavas together with lacustrine and fluviatile deposits. The surface of the old metamorphic complex decreases in elevation southward and is covered with an enormous thickness of Tertiary lavas. These have been deeply cut into by the Columbia River along the Oregon-Washington boundary line. At this point the surface of the old metamorphic complex is presumably below sea level. Throughout the entire Cascade Range, all the igneous formations may be regarded as totally barren of oil deposits. The same may be said regarding the pre-Tertiary metamorphics. It is barely possible that certain portions of these older formations may have escaped metamorphism and might possibly contain oil, although there is absolutely no evidence to suggest that such deposits are present. On the eastern slopes of the Cascade Mountains there are extensive areas of Eocene sandstones and shales containing coal

seams. These sediments are presumably of lacustrine origin. No organic remains have been found within them to suggest a possible source for petroleum. From such knowledge as we have concerning the geology of the Cascade Mountains it seems fairly safe to assume that the conditions are adverse to the formation or accumulation of petroleum deposits.

In the Puget Sound Basin, southwestern Washington, and in the Olympic Peninsula the geologic conditions are entirely different from those obtaining in the Cascades or in eastern Washington. The Tertiary formations of western Washington consist of sandstones and shales of marine, brackish-water and fresh-water origin. They were formed during the Eocene and Miocene periods, in embayments of the ocean or in estuaries not far from the shore line. The fossil remains of marine animal life occur in these strata. Intercalated with the Eocene beds of both marine and brackish-water origin are basaltic flows. All of these strata have been subjected to deformational movements and are now involved in anticlinal and synclinal folds. Later these strata were subjected to intensive erosion and many of the folds have been deeply cut into. During the Pleistocene, these rocks were heavily veneered with thick deposits of glacial drift.

The Eocene formations consist of 10,000 or 12,000 ft. of sedimentary strata and intercalated basalts. In King and Pierce Counties and other portions of the northern Puget Sound Basin these strata are almost entirely of estuarine origin. They contain numerous interbedded layers of carbonaceous seams but very rarely any deposits of animal origin. In southwestern Washington marine sediments prevail, although brackish-water bands are found interbedded. In the lower portion of the Eocene, the igneous flows prevail. However, marine and estuarine lenses are intercalated. The Miocene strata in both southwestern Washington and in the Puget Sound Basin attain a maximum thickness of 10,000 ft. Along the northern border of the Olympic Mountains the Miocene is nearly 19,000 ft. thick. It rests unconformably upon the Eocene basalts and shales, which in turn are lying unconformably upon the upturned edges of the older Mesozoic rocks of the Olympic Mountains. The Tertiary formations in the western part of the State are being prospected for possible deposits of petroleum. What are the geological conditions which might warrant the supposition that petroleum deposits are present in the Tertiary formations? The larger portion of the Tertiary formations in the Puget Sound Basin are thickly covered with glacial drift. Occasional exposures of bed-rock project up through the drift. It is from such exposures that the character and structure of the older rocks must be worked out. A prominent spur from the Cascade Mountains extends westerly into Kitsap County. The Eocene rocks are composed of brackish-water

sediments with intercalated basalts. Resting unconformably upon these are 10,000 ft. of lower Miocene shales and sandstones of marine origin. The core of this anticline has been subjected to such extensive erosion that the Miocene strata which formerly arched over the axis of the anticline have been completely removed. The truncated edges of the Miocene strata, along the entire course of the anticline, are exposed to the surface except where covered with drift. To the north a broad synclinal basin exists and the Tertiary rocks are covered with an enormous accumulation of glacial deposit. To the south of this anticlinal axis no exposures of the Miocene are known to exist until the Chehalis River is reached. The Eocene basalts cannot possibly have been a source for petroleum. Such sedimentary strata as are intercalated with the basalts are of brackish-water origin and could not have acted as a source for oil. The overlying Miocene strata contain marine fossils which might possibly at one time have acted as a source for petroleum. Assuming that such deposits may at one time have been formed, it would be impossible for them to be preserved to-day inasmuch as the strata along the axis of the anticlinal fold which might have retained them have been completely removed by erosion. If boring were undertaken in Seattle or to the north it would be necessary to penetrate an enormous thickness of drift, and when the Miocene strata were reached they would be found to exist close to the axis of a synclinal trough. The geological conditions in the Puget Sound Basin point very strongly to the fact that commercial deposits of petroleum cannot be expected to occur in this region.

From the southern portion of the Puget Sound Basin south to the Cowlitz Basin the Eocene formations have been thrown into shallow folds with axes having a prevailing trend from northwest to southeast. The marine phases of the strata contain organic remains which might possibly have acted as a source for small amounts of petroleum. No surface indications to denote the occurrence of petroleum are definitely known to exist. There is no geological information present to warrant the statement that petroleum deposits cannot occur in Thurston, western Lewis, and Cowlitz Counties. Neither are there facts to suggest the occurrence of commercial deposits. The probabilities are that if they should be found to be present it would only be in comparatively small amounts, as no extensive remains of organic material occur which could have acted as a source of origin.

The southwestern portion of the State contains Eocene and Miocene strata with associated marine organic deposits. These have been folded. It is possible that oil may be present in small quantities beneath the anticlines. No definite seepages have been recorded except in a small area on Bear River near the mouth of Columbia River. In pros-

pecting this region bore holes should be sunk upon the axes of the anticlines and the venture should be regarded as purely experimental.

On the northern flank of the Olympic Mountains an extensive series of lower Miocene formations of sedimentary origin occur. These deposits consist exclusively of shales and subordinate amounts of sandstone and conglomerate. They are entirely marine and in places are fossiliferous. They have been folded and to a certain extent eroded. In most places they are mantled with glacial drift. No direct indications of oil have been seen in this region although there is no evidence to prove conclusively that it is not present. Provided bore holes were sunk upon the axes of the anticlines drilling would be warranted from an experimental standpoint.

On the western border of the Olympic Peninsula there is present a sedimentary formation possessing a thickness of at least 10,000 ft. which is certainly pre-Tertiary in age. This assemblage of strata is termed the Hoh formation. It is impossible absolutely to determine its age. It may be Cretaceous, Jurassic, or even older. In the northern Cascades and on Vancouver Island the Triassic and Carboniferous formations are metamorphosed. The Hoh formation is altered to only a very minor degree. It is much more indurated than the Chico or upper Cretaceous as exposed on the northeastern flank of Vancouver Island. It resembles in lithologic appearance somewhat the Jurassic rocks on Vancouver Island. However, lithologic characteristics form a poor basis for correlation. This formation extends from Point of Arches, just south of Cape Flattery, southward to Queets River and from the Pacific Ocean easterly in to the center of the Olympic Range for an undetermined distance. The formation is composed predominately of dark-gray shales and sandy micaceous shales with subordinate amounts of gray medium-grained sandstones which in places are gritty. Occasionally bands of conglomerate are present. Some of the shale bands are often composed of alternating belts of thinly bedded sandstones and shales the individual members of which are only a few inches in thickness. This entire series of strata have been thrown into anticlinal and synclinal folds. Two of these folds trend nearly north and south parallel to the coast. On the flanks of these major folds minor undulatory folds or warps have been developed. All of these structures have been deeply dissected by erosion and upon the eroded surface great deposits of fluviatile sands and gravels have been laid down. Later these have been cut into by the present rivers flowing westerly to the coast. Along the banks beneath the overlying gravels the strata of the Hoh formation are exposed in places. Good exposures may also be seen in the cliffs along the ocean. No fossils have as yet been found in this formation with the exception of fragments of carbonized wood, and the remains of diatoms and foraminifera. No traces of molluscan

remains are known to occur. At a number of localities within this formation small seepages of oil may be seen in the shaly sandstones. At many other localities a pronounced odor of petroleum may be detected even at some distance from the rock exposure. About 2 miles north of the mouth of Hoh River in western Jefferson County and just south of Hoh Head a seepage occurs on the beach. One-fourth mile inland the odor is extremely noticeable. Several pits have been made and a shaft has been sunk to a depth of 20 ft. In the bottom of the shaft a small amount of oil seeps out. This has been collected from time to time and several analyses have been made. The results show the oil to have a paraffin base and low specific gravity. About 7 miles to the east another seepage occurs. Along the coast northward to Quillayute River localities where the odor of oil is pronounced are numerous. At a number of places along Bogachiel River similar exposures may be seen. It was the presence of these indications which originally gave incentive to the search for petroleum in commercial quantities. In 1901 and 1902 an attempt was made to drill for oil south of the mouth of Quillayute River on the coast. The efforts were unsuccessful. Apparently no attention was given to the structural conditions, as the well is located upon the flank of a syncline. In 1912, a well was started by the Washington Oil Co. at a point 1 mile south of Forks Prairie in Clallam County. At the present time the well is said to be at a depth of 2,100 ft. The well is being sunk upon the axis of a well-defined anticline trending nearly north and south. Considerable amounts of gas were encountered at several depths.

Early in the following year a well was started by the Jefferson Oil Co., near the oil seep previously mentioned as occurring near Hoh Head. This well is being sunk on a short anticlinal fold trending northeast to southwest. The rock encountered to the present depth is mostly shale with alternating beds of sandy shale.

Farther south, just north of the mouth of Queniult River a well is being drilled close to a gas mound. The rock formation at the surface is upper Miocene. A low monoclinal fold trends northeast from the coast. Above it on the surface of the ridge about 300 ft. above sea level a small mound about 200 ft. in diameter exists. On the apex there is a small crater-like opening filled with water. Out of this gas is almost continuously escaping. Presumably the source is from the Hoh formation, which lies unconformably below the upper Miocene.

It is possible that the Hoh formation extends into the heart of the Olympic Range. Reports are current of the presence of similar oil indications in that region. Whether petroleum deposits occur in commercial quantities or not in the Hoh formation has not as yet been demonstrated. The indications are sufficient to warrant a very careful examination and a series of test holes, provided they are sunk upon the

axes of the anticlinal folds. There is absolutely no other possible means of predicting whether oil will be found in commercial quantities.

The origin of such petroleum as is known to be present cannot be absolutely proved. At a depth of 1,800 ft. in the Forks well, samples of shale show the presence of numerous skeletons of foraminifera. It is quite possible that these forms of organic life may have in part been the source. The shales near the Hoh River outcrops contain remains of diatoms. Some specimens of this shale upon microscopic examination are found to be crowded with diatoms. The soft organic portions of living diatoms are known to have fats as the organic constituents rather than cellulose. In the California districts diatomaceous shales are believed in many cases to have been the source of the oils. The shales bearing diatoms in the Hoh River region are not siliceous as in California. They compose only a small proportion of the rock. It is quite possible that the diatoms may also in part have contributed as a source for the oil. If oil deposits are ultimately found to be present in commercial quantities they may be expected to exist in the more porous sandstone zones. Such zones are present within the formation, as well as thick shale belts which can act as impervious cappings. Provided oil deposits are present, they would not necessarily exist beneath the areas where seepages occur upon the surface. Fractures in the strata might conduct such emanations of oil or gas to the surface over the axis of a syncline, which would be a very unfavorable place to drill. If oil is present in commercial quantities there is no possible means at present of predicting at what depths oil-bearing zones may be expected to occur. The same zone is certain to vary in depth at different localities.

Conclusions

From the geological conditions known to exist in the Cascade Mountains and in eastern Washington the chances for the occurrence of petroleum deposits in commercial quantities are very unfavorable. The Eocene formations in the Puget Sound Basin have been formed under conditions which would not allow organic material to accumulate as a source of petroleum. The overlying lower Miocene strata which were involved in a great anticlinal fold have been so extensively eroded that even though at one time oil deposits may have existed within them they would long since have been destroyed as the result of erosion. Conditions are thought to be unfavorable for the occurrence of petroleum deposits in the Puget Sound Basin.

In Thurston, western Lewis, Cowlitz, Pacific and southern Chehalis Counties, Eocene and Miocene formations are present which are in part of marine origin and contain considerable quantities of marine fossils. Both shales and sandstones exist which could act as retainers

and cappings. They have been folded into shallow folds. No seepages or direct indications of the presence of petroleum are known to occur. There are no facts which would prohibit its occurrence and none to suggest its presence. Several wells are being bored in this region.

The only definite indications of the presence of petroleum within the State occur on the western side of the Olympic Peninsula in the Hoh formation. This formation is possibly Jurassic in age. The formation is about 10,000 ft. thick and has been folded into anticlines and synclines. At several places seepages of oil and emanations of gas are known to occur. The source of such oil as is known to occur may be derived from the remains of diatoms and foraminifera. Three wells are being drilled, the deepest of which is at a depth of 2,100 ft. Oil in small amounts definitely occurs, but whether it is present in commercial quantities or not can only be determined by drilling in those localities where structural conditions would permit of its occurrence.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Formation and Distribution of Bog Iron-Ore Deposits

BY C. L. DAKE,* M. A., ROLLA, MO.

(San Francisco Meeting, September, 1915)

METHOD OF FORMATION OF BOG ORES

Chemistry of Iron Solution

IRON is much more soluble in the ferrous than in the ferric form. Where, as in the case of the ferrous silicates and the sulphides, the iron is already in the ferrous form, it may go at once into solution and be removed, upon the breaking down of the parent rock. If in the ferric form, it must first be reduced; usually by the decay of organic matter, the demand of the decaying carbonaceous matter for oxygen being so great that the latter is abstracted from the ferric compounds and the iron is thereby reduced and rendered soluble. The iron may go into solution as the sulphate, or in the presence of excess CO_2 as ferrous carbonate. It is probably soluble as salts of the organic acids.¹ Recent work indicates that these are important in the formation of bog ores.² There is considerable variety of opinion concerning the behavior of these organic compounds.³

The secretions about the roots of plants and trees also cause the solution and removal of iron and a consequent bleaching of the soil.⁴

While the matter is not yet well understood, we can probably say definitely that the so-called organic acids are a group of colloids which possess slightly acid properties, and the salts of which are partly soluble

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¹ Beck, Richard: *The Nature of Ore Deposits*, vol. i, pp. 101, 102 (1905.)

Julien, A. A.: Geological Action of the Humus Acids, *Proceedings of the American Association for the Advancement of Science*, vol. xxviii, p. 311 (1879).

² Aschan, Ossian: Die Bedeutung der wasserlöslichen Humusstoffe für die Bildung der See- und Sumpferze, *Zeitschrift für praktische Geologie*, vol. xv, No. 2, p. 56 (Feb., 1907).

³ Aschan, Ossian: *Loc. cit.*

Julien, A. A.: *Loc. cit.*

Hilgard, E. W.: *Soils*, pp. 122, 123 (1906).

⁴ Phillips, J. A., and Louis, H.: *A Treatise on Ore Deposits*, p. 36 (1896).

and partly insoluble.⁵ Their iron salts readily change from the ferrous to the ferric form, and their solubility is greatly modified by the presence of ammonia.

Chemistry of Iron Precipitation

The simplest cause of precipitation of iron is oxidation from the soluble ferrous to the much less soluble ferric compounds. Ferrous carbonate, soluble in carbonated waters, is also precipitated as the carbonate by evaporation of the excess CO₂. Certain bacteria precipitate iron by building the hydrated oxide into their body cells.⁶ Iron is precipitated as the sulphide when vegetation decays in the presence of sulphates of lime and magnesia, with a deficiency of oxygen.⁷ There are many other means⁸ by which, under special conditions, iron may be precipitated in nature. Oxidation, evaporation of CO₂, and iron bacteria have usually been held to be most important.

CHARACTER OF BOG ORES

Bogs are often accompanied by thin layers of iron ore, consisting largely of hydrated ferric oxide, frequently with considerable amounts of ferrous carbonate, and sometimes ferrous sulphate and silicate.⁹ In marine marshes, under the influence of the sulphates of sea water, iron may go down as the sulphide.¹⁰ Bog ores, however, consist essentially of limonite, with subordinate amounts of carbonate. Such ores are usually high in phosphorus¹¹ and almost invariably preserve abundant fossil fragments of leaves and plant remains, which serve to identify them as of bog origin.¹¹

⁵ Clarke, F. W.: *The Data of Geochemistry, Bulletin No. 491, U. S. Geological Survey*, p. 98 (1911).

⁶ Winchell, N. H. and H. V.: *The Iron Ores of Minnesota, Bulletin No. 6, Minnesota Geological and Natural History Survey*, p. 223.

Beck, Richard: *The Nature of Ore Deposits*, vol. i, pp. 101, 102 (1905).

Bischof, Gustav: *Elements of Chemical and Physical Geology*, vol. i, p. 169 (1854).

⁷ Dawson, Sir William: *Geology of Nova Scotia*, p. 24.

Geikie, Sir A.: *Text-Book of Geology*, 4th ed., p. 612 (1903).

Bischof, Gustav: *Op. cit.*, p. 162.

⁸ Kemp, J. F.: *Ore Deposits of the United States and Canada*, 5th ed., p. 87 (1903).

Beck, Richard: *Loc. cit.*

⁹ Moore, E. J.: *Occurrence and Origin of Some Bog Iron Deposits in the District of Thunder Bay, Ontario, Economic Geology*, vol. v, No. 6, p. 528 (Sept., 1910).

¹⁰ Dawson, Sir William: *Loc. cit.*

¹¹ Winchell, N. H. and H. V.: *Op. cit.*, p. 221.

Phillips, J. A., and Louis, H.: *Op. cit.*, p. 42.

Kemp, J. F.: *Op. cit.*, p. 89.

DISTRIBUTION OF BOG ORES

Relations to Geographical Latitude

Those countries best known as producers of bog ores are Norway, Sweden, Germany, Poland, Finland, and Canada.¹² Many deposits occur in the northern United States.¹³ A few localities are known in regions farther south, among them the deposits of Missouri,¹⁴ North Carolina,¹⁵ and Rio Tinto, Spain.¹⁶ Exhaustive lists of bog-ore districts, however, include few localities outside the limits of the regions just named. Careful examination has shown that the overwhelmingly larger number of the known bog deposits occur in the northern portions of the three northern continents.

Relations to Glaciation

A very small percentage of our bog-ore deposits is to be found outside the glaciated regions, while within those regions they are characteristic and abundant, both in Europe and in North America.¹⁷ The ores accumulate in the smaller swamps, while in large neighboring morasses they may be entirely wanting, probably by reason of the fact that the concentration is greater in the smaller swamps.¹⁸ Glaciated areas are more favorable to iron accumulation, because they present numerous small swamps, rather than the immense ones, such as are common to the lower flood plains and deltas of large rivers, or the saline marshes of low coastal plains.

Relation to Sources of Supply

Another factor, doubtless, is the proximity of available sources of supply for the iron. From 1 to 3 per cent. of the glacial drift is said to be made up of magnetite;¹⁹ and other iron compounds are present. The unconsolidated condition of the drift renders this iron content easily

¹² Roth, Justus: *Allgemeine und chemische Geologie*, vol. i, pp. 597, 598, 645.

Walther, J.: *Einleitung in die Geologie*, p. 751.

Beck, Richard: *Op. cit.*, p. 100.

Bischof, Gustav: *Loc. cit.*

Phillips and Louis: *Op. cit.*, p. 37.

¹³ Kemp, J. F.: *Op. cit.*, pp. 89, 90.

¹⁴ Nason, F. L.: Iron Ores of Missouri, *Missouri Geological Survey*, vol. ii, p. 182 (1892).

¹⁵ Nitze, H. B. C.: Iron Ores of North Carolina, *Bulletin No. 1, North Carolina Geological Survey* (1893).

¹⁶ Phillips and Louis: *Op. cit.*, pp. 40 to 42.

¹⁷ Shaler, N. S.: General Account of the Fresh-Water Morasses of the United States, *Tenth Annual Report, U. S. Geological Survey*, pt. 1, p. 305 (1888-89).

¹⁸ *Op. cit.*, p. 306.

¹⁹ *Op. cit.*, p. 305.

accessible. The glaciated regions parallel somewhat closely the so-called "Pre-Cambrian Shields" of both Europe and North America, areas in which enormous amounts of older iron-rich rocks are localized. This may also account in part for the noteworthy supply of iron for bog deposits in both drift and bed rock of the glaciated regions.

Of the localities of bog ores cited outside the glaciated area, three at least are definitely related to local sources of iron. In the case of the Rio Tinto beds the conditions are as follows:²⁰ A marsh or shallow lake extended over the area where the bog ores are now found, and into this flowed iron solutions resulting from the decomposition of extensive beds of pyrites. From these chalybeate waters, the bog ores were, and are still, thrown down. In Park County, Colo.,²¹ water which percolates through the rock is charged with iron sulphate, resulting from the solution of beds of pyrites in the vicinity. The iron is thrown down in a small marsh, through which the drainage of the region passes. In the Missouri deposits, a very ferruginous sandstone is underlain by an impervious shale. Bogs form along the contact where the percolating waters escape, and here the ores are deposited.²² From the above three cases, it will be seen that an immediate source of iron may become one of the factors in localizing such deposits.

Relation to Climate and Its Chemical Effect

The prevailing red color of soils in the tropics has been assumed to be a result of a higher oxidation of the iron and its superficial dehydration. It seems possible that the rapid decay of vegetation in tropical regions, and the resulting lack of accumulation of partly decayed vegetable matter, may effect the accumulation of bog iron ores.

Two alternative suggestions have been tentatively advanced: first, that iron does not go into solution in the tropics as abundantly as in colder regions; and secondly, that it goes into solution abundantly, but is not as readily precipitated.

Iron Solvents in Tropical Soils.—Earth with a high content of partly decayed vegetation is found to relatively slighter extent in the tropics than elsewhere, because the conditions which favor rapid growth favor rapid decay also. The lack of large quantities of humus in the tropical soils, except at high altitudes,²³ is pretty generally reported. Peat

²⁰ Phillips and Louis: *Op. cit.*, pp. 40 to 42.

²¹ *Loc. cit.*

²² Nason, F. L.: *Op. cit.*, p. 182.

²³ Taylor, A. E.: *The Peat Deposits of Indiana, Thirty-first Annual Report, Department of Geology and Natural Resources, Indiana*, p. 75 (1906).

Walther, J.: *Einleitung in die Geologie*, pp. 745, 752, 811.

Roth, Justus: *Allgemeine und Chemische Geologie*, vol. ii, p. 645.

Schimper, A. F. W.: *Plant Geography on a Physiological Basis*, p. 380 (1903)

accumulates only where there is a nearly permanent water supply, and areas of accumulation are few outside the glaciated region, by reason of high temperature, long summers, frequent drought, and lack of undrained areas.²⁴ It must be borne in mind, however, that the accumulation of partly decayed woody matter is not necessarily indispensable to the development of abundant organic acids. In this connection Hilgard,²⁵ somewhat in opposition to the general conception, concludes that tropical soils are likely to be rich in humus, where the interval between the rainy seasons is not too long; otherwise, an almost complete oxidation of the humus must take place, leaving only a trifling organic residue. Organic acids are drained away to such an extent that during the rainy season many tropical rivers become deep coffee brown.²⁶

The evidence seems to be somewhat contradictory. Where tropical soils are subjected to long and frequent drought, humus is rare. In tropical regions of nearly continuous heavy rainfall, humus is said to be abundant. Excessive decay must yield plentiful CO₂. Growing roots must yield abundant solvents. Tropical soils, then, may or may not carry iron solvents, the distribution depending largely on drought and rainfall.

Iron Solvents in Tropical Rivers.—Many tributaries of the Amazon and Orinoco are colored nearly black by organic acids.²⁷ The following tabulation gives for several well-known rivers the amount of organic matter.²⁸

River	Organic Matter, Per Cent.	Salinity, Parts per Million
Danube.....	3.25	184
James.....	4.14	89
Nile.....	10.36	119
Hudson.....	11.42	108
Cumberland.....	12.08	124
Amazon.....	15.03	59
Delaware.....	16.00	70
Xingu.....	20.63	45
Tapajos.....	24.16	38
Plata.....	49.59	91 to 206
Negro.....	53.89	132
Uruguay.....	59.90	66

²⁴ Davis, C. A.: Preliminary Report on the Peat Deposits of North Carolina, *Economic Paper No. 15, North Carolina Geological and Economic Survey*, pp. 147, 148 (1908).

²⁵ Hilgard, E. W.: *Soils*, pp. 129, 399 (1906).

²⁶ Schimper, A. F. W.: *Loc. cit.*

²⁷ Clarke, F. W.: The Data of Geochemistry, *Bulletin No. 491, U. S. Geological Survey*, p. 82 (1911).

²⁸ *Op. cit.*, p. 97.

There is evidently a decided tendency in tropical rivers to carry a higher percentage of organic matter than do those of temperate zones. Their total salinity, however, is rather lower. We have no data as to the relative amounts of dissolved CO_2 carried by tropical and temperate rivers, but it is believed that the abundant decaying vegetation in tropical waters would yield much CO_2 . We may then safely say that solvents for iron are not lacking in tropical streams.

Iron Content of Tropical Rivers.—In only a few analyses of tropical waters have separate determinations of Fe_2O_3 and Al_2O_3 been made.

Analyses of Tropical River Waters.²⁹

River	Fe_2O_3 Parts per Million	Total Salinity Parts per Million
Plata	4.0131	91
Parana	3.1458	98
Nile	3.1535	119

Analyses of Temperate River Waters for Comparison.²⁹

Mississippi at New Orleans	0.2158	166
St. Lawrence at Ogdensburg	0.0804	134
Danube near Vienna	0.3340	167

The three tropical rivers in the above list are trunk streams, from which the water was taken, well down toward the mouth. As will be noted, they run far higher in iron than the three streams in temperate regions cited for comparison. The analyses given are too few in number to warrant the positive statement that tropical rivers in general carry a high iron content; but the figures are suggestive.

The Precipitation of Iron in Tropical Regions.—It has already been shown that bacteria aid in the precipitation of iron. Whether tropical conditions increase or decrease the activity of such bacteria is unknown.

It is possible that the high content of the rivers in solvents may tend to prevent precipitation. I know of no careful determination of organic matter or dissolved CO_2 in the waters of those bogs in which iron is now going down. In Scandinavian lakes where lake ores are forming, there is a marked content of humic acids;³⁰ and Lough Neagh, in Ireland, which has a high organic content,³¹ carries also 10.416 parts per million of Fe_2O_3 .³² In the latter case, the water seems to be high both in iron and in organic matter; but I have seen no positive statement that bog-ore

²⁹ *Op. cit.*, pp. 63 to 97.

³⁰ Aschan, Ossian: Die Bedeutung der wasserlöslichen Humusstoffe für die Bildung der See-und Sumpferze, *Zeitschrift für praktische Geologie*, vol. xv, No. 2, p. 56 (Feb., 1907).

³¹ Clarke, F. W.: *Op. cit.*, p. 97.

³² *Op. cit.*, p. 85.

deposits are actually forming in Lough Neagh at the present time, although the proportion of iron is sufficient to impregnate or "petrify" wood placed in the lake.³³ If the organic matter of bogs where iron is now depositing is as high as that of tropical rivers (as seems to be suggested above), this organic content cannot be adduced as a determining factor in precipitation. More evidence is needed on this point.

Partial Explanation.—Most tropical soils, by reason of long, hot, dry seasons, are deficient in humus, and contain iron in a high state of oxidation, and therefore relatively insoluble. But regions of nearly continuous heavy rainfall are covered with rank vegetation, and rapid decay yields plentiful humus. In such regions, therefore, iron may go into solution freely; but the abundant rains will cause excessive leaching of the humus, removing the dissolved iron. Thus tropical soils may, on the average, be low in humus, while the rivers carry, nevertheless, much organic matter, and hold iron in solution. The high state of oxidation of the iron as a whole, together with the extensive leaching of such iron as is soluble, and its rapid removal in waters with a high solvent power, together with the absence of small swamps giving opportunity for concentration, may possibly furnish a partial explanation of the infrequent occurrence of bog ores in the tropics.

Marine Bog Ores

In reviewing the literature of bog deposits, I have found only three references to ores now forming in marine marshes. In the tidal marshes of Nova Scotia iron sulphide is being formed.³⁴ No data are given concerning the extent of the deposits. Geikie³⁵ speaks of extensive marshes in the eastern United States in which iron is being deposited. Unfortunately he does not cite a definite locality, or give references for his authority. At Rio Tinto the mine waters are carrying iron to Huelva Bay, but no mention is made of the extent of the deposits.³⁶

In none of the above descriptions do we find well-authenticated examples of typical bog ores now forming in marine marshes. Several articles on littoral sediments and sea-coast swamps³⁷ discuss in much

³³ Bischof, Gustav: *Elements of Chemical and Physical Geology*, vol. i, p. 95 (1854).

³⁴ Dawson, Sir William: *Geology of Nova Scotia*, p. 24.

³⁵ Geikie, Sir A.: *Text-Book of Geology*, 4th ed., p. 612 (1903).

³⁶ Beck, Richard: *The Nature of Ore Deposits*, vol. i, p. 103 (1905).

³⁷ Delesse, A.: *Lithology of the Seas of the Old World*, *Quarterly Journal of the Geological Society*, vol. xxvi, pt. 2, p. 5 (1870).

Delesse, A.: *Lithologie du Fond des Mers*.

Delesse, A.: *Lithology of the Seas of the New World*, *Quarterly Journal of the Geological Society*, vol. xxviii, pt. 2, p. 1 (1872).

Shaler, N. S.: *Sea-Coast Swamps of the Eastern United States*, *Sixth Annual Report, U. S. Geological Survey*, p. 359 (1884-85).

detail the nature of such deposits, but nowhere in these articles is mention made of iron deposits now forming. Beck²⁸ concludes that aside from the deposits of iron forming at present in Huelva Bay, no really recent iron ores are known on the sea bottom.

Lake Ores

Lake ores, described as occurring chiefly in Sweden, Finland, European Russia, and Canada,²⁹ may be classed with bog ores, since they are essentially the same in distribution, method of formation, and character, and frequently show gradations into bog ores.

SUMMARY

Bog ores are widely distributed, but occur chiefly within the colder temperate regions, and for the most part within the glaciated area. No important deposits are known to be forming in marine bogs.

Causes of the Above Distribution

The probable causes of this distribution are:

1. Swamps are more numerous in the glaciated area than elsewhere.
2. Glacial swamps are of smaller size and allow greater concentration of chalybeate waters.
3. There is abundant available iron in glacial soils.
4. Possible climatic factors involve the high state of oxidation of the iron in warmer regions, together with the lack of iron solvents in large areas of tropical soils, and abundant organic matter in many of the rivers keeping in solution the iron once dissolved, and permitting it to be carried out to sea, instead of being re-deposited in swamps.

The writer believes that glaciation is probably the most important factor, operating through the favorable conditions presented by the presence of numerous small swamps, and the abundance of iron available in glacial soils.

So far as climatic factors are concerned, not enough evidence is available to warrant definite conclusions; but certain possibilities have been pointed out, and further lines of research have been suggested.

²⁸ Beck, Richard: *Loc. cit.*

²⁹ *Op. cit.*, p. 100.

Biographical Notice of John Birkinbine

BY ROSSITER W. RAYMOND, NEW YORK, N. Y.

JOHN BIRKINBINE was born Nov. 16, 1844, at Reading, Pa., the eldest son of H. P. M. Birkinbine, widely known as a hydraulic engineer. The family removed subsequently to Philadelphia, where, as a young man he established with his father the office now continued by his sons, after more than 60 years.

His education was received at public schools, the Friends' High School in Philadelphia, the Hill School at Pottstown, Pa., and the Polytechnic College of Pennsylvania, where his studies were interrupted in 1863-4 by service in the Union Army, which included participation in the battles at and around Gettysburg. Later, he devoted two years to work in a machine shop; and subsequently he became associated with the late P. L. Weimer, under the firm name of Weimer & Birkinbine, operating the Weimer Machine Works at Lebanon, Pa.

Much of his work was in mining, metallurgy, and blast-furnace construction. As manager for the South Mountain Mining & Iron Co. he carried on experiments with various fuels for iron-ore smelting while maintaining the furnace in constant operation. The carefully recorded results obtained were widely published, and are referred to in text-books by other metallurgists as most complete.

From his Philadelphia office he was sent to nearly every State, and to Canada and Mexico, for examinations, reports, constructions or improvements in iron-ore mines, blast furnaces, iron works, water supplies, hydraulic development, irrigation projects, etc., and his engineering knowledge was requisitioned by several European corporations. A number of business trips were made to Mexico, beginning with a visit to the Cerro de Mercado at Durango, before railroads were established in that portion of Mexico, to make a critical examination and report on this "Iron Mountain." Later visits covered other localities and engineering problems, familiarizing him with the major part of the iron industry in Mexico. The late disturbed political conditions in that Republic have retarded the probable enlargement, modernization, and improvement of much of the iron and steel industry of Mexico, upon which he investigated and reported for various capitalists on both continents. One interesting subject included in these reports was a proposed electric furnace, to be operated by energy from water power.

Mr. Birkinbine was one of the first to suggest an iron industry at the head of the Great Lakes, using coke made from Pennsylvania coal. His report was an important factor in establishing the iron industry at the head of Lake Superior; and the blast furnace at West Duluth, Minn.,



JOHN BIRKINBINE.

was built under his plans and supervision. He was engaged by the State of Texas to investigate the practicability of iron manufacture in that State. As an engineer he co-operated with E. S. Cook of Pottstown, Pa., who did much to advance the iron blast-furnace industry. He was for

some years Consulting Engineer for the Philadelphia & Reading Coal & Iron Co., and held a similar position with Thomas A. Edison during the latter's early experiments in magnetic concentration of iron ore, and with Witherbee, Sherman & Co. in beneficiation tests. Also for the Colorado Fuel & Iron Co. in the enlargement and improvement of their works and the construction of an augmented water-supply system.

In his reports and recommendations, his conclusions were clearly stated, and a reputation for conservatism and fairness brought him numerous cases of valuation, adjustment, and arbitration, in some of which, by mutual consent, he was the representative of both parties.

He always adhered to the policy of accepting no financial interest or contingent fees whatever, and would patent none of his numerous improvements or ideas, so that personal bias in his statements or conclusions could not even be suggested.

He also acted as an expert adviser for investing capitalists, and for a number of the greatest industrial corporations and several large railroad companies in this country. He was Chief Engineer, Vice-President, and Chairman of the Committee of Awards of the National Export Exposition, served on Juries of Awards at the Centennial, World's Columbian, Pan-American, and Cotton States General Expositions, and was named for similar duties at others.

From its inception in 1905 he was Chairman of the Water-Supply Commission of Pennsylvania, patriotically devoting to this work, for a nominal recompense, a large portion of his valuable time. As a result, he established an efficient organization, not only free from political influence, but noted for the zeal and faithfulness with which each member performed his duties.

He was active in forming the Pennsylvania Forestry Association, the largest and most influential of its class, and was its President for 23 years, during which time the Association accomplished the appointment of a State Forestry Commission (later made a State Department) and the enactment of statutes which encouraged the forestry movement. From the establishment of this Association, he edited its publications.

He was also active in the formation of the United States Association of Charcoal Iron Workers, of which he was Secretary, and for nine years the editor of its journal. For many years he was Special Agent for the United States Geological Survey, preparing the reports on Iron Ores for the 11th and 12th Censuses, and that on Manganese Ores for the 12th Census, and has since prepared for the Survey additional data and studies. He was appointed by the Secretary of the Interior Expert Metallurgical Engineer for the Bureau of Mines. He received marks of approval from the Survey and from several foreign scientific societies, and was member of a number of international congresses.

For 10 years he served as President of the Franklin Institute. He was

also a member of the American Society of Mechanical Engineers, the Engineers' Club of New York, the American Society for Testing Materials, the Engineers' Club of Philadelphia (President in 1893), the Manufacturers' Club of Philadelphia, the Pennsylvania Foundrymen's Association, the George G. Meade Post No. 1, G. A. R., of Philadelphia, and was an Honorary Member of the Canadian Mining Institute.

Mr. Birkinbine refused honorary degrees from two colleges, modestly saying that as he had been unable to graduate from his own *Alma Mater* he was not warranted in accepting a higher degree.

During his career Mr. Birkinbine also maintained his specialty of hydraulic engineering, acting as engineer on water supplies for various municipalities. He not only witnessed, but had active participation in, the development of water power for electrical energy. While he was at college, electricity was a laboratory experiment only, and its first exhibition as an illuminant was at Philadelphia about 12 years later; while the use of water power was then confined to limited volumes at low heads for direct mechanical purposes. His activities covered the development of hydro-electric science to its present advanced stage. In 1888 he prepared a comprehensive report on the development of the great water power of the St. Louis River in Minnesota, considering a 15-mile transmission, though no water-wheel manufacturer would guarantee turbines for heads above 35 feet. Since then he was associated with or reported on many developments in various States and in Mexico, covering high heads or large volumes of water until lately deemed impracticable.

Mr. Birkinbine became a member of this Institute in 1875, a Manager in 1883, Vice-President in 1887, and President in 1891 and 1892. The following is a list of his contributions to the *Transactions*:

Papers

Title	Vol.	Page	Year
Suspended Hot-Blast Stoves.....	IV	208	1875
Pumping Engines.....	V	455	1876
Notes upon the Drainage of a Flooded Ore-Pit at Pine Grove Furnace, Pa.....	VI	174	1878
The Production of Charcoal for Iron Works.....	VII	149	1878
Experiments with Charcoal, Coke and Anthracite in the Pine Grove Furnace, Pa.....	VIII	168	1879
A Short Blast at the Warwick Furnace, Pennsylvania.....	IX	51	1880
Charcoal as a Fuel for Metallurgical Processes.....	XI	78	1882
Roasting Iron-Ores.....	XII	361	1883
The Cerro de Mercado (Iron Mountain) at Durango, Mexico.....	XIII	189	1884
The Distribution and Proportions of American Blast-Furnaces.....	XIV	561	1885
Operation of Warwick Furnace, Pennsylvania, from August 27th, 1880, to September 1st, 1885.....	XIV	833	1886

Papers.—Continued

Title	Vol.	Page	Year
Comparisons of Blast-Furnace Records.....	XV	147	1886
A Tilting-Ladle Car for Molten Metal or Slag.....	XV	685	1887
The Distribution and Proportions of American Blast-Furnaces. (Second Paper.).....	XV	690	1887
The Resources of the Lake Superior Region.....	XVI	168	1887
Prominent Sources of Iron-Ore Supply.....	XVII	715	1889
Crystalline Magnetite in the Port Henry, New York, Mines.....	XVIII	747	1890
Progress in Magnetic Concentration of Iron-Ore.....	XIX	656	1890
The Fuel-Supply of the United States. A Sketch of the Progress of Twenty Years in the Economy of Production and Consumption. (Presidential Address.).....	XX	409	1891
The Influence of Location upon the Pig-Iron Industry. (Presidential Address.).....	XXI	473	1892
Industries of the Schuylkill Valley. (Presidential Address.).....	XXI	618	1892
The Development of Technical Societies. (Presidential Address.).....	XXI	962	1893
Note on a Supposed Aztec Mirror.....	XXIV	617	1894
Note on a Piece of Carpenter Steel.....	XXIV	619	1894
The Iron-Ore Supply.....	XXVII	519	1897
Distribution of the World's Production of Pig-Iron.....	XXX	504	1900
Hydraulic Pumping-Plant on the Snake River, Idaho, for Power, Irrigation and the Treatment of Gold-Sands.....	XXX	518	1900
Growth of the Pig-Iron Production During the Past Thirty Years. (Not Printed.).....	XXXIII	xxxvi	1902
Biographical Notice of William George Neilson.....	XXXVIII	402	1907
The American Institute of Mining Engineers and the Conservation of Natural Resources.....	XL	412	1909

Remarks in Discussion

Blast-Furnace Hearths and In-Walls.....	IV	186	1875
The Economy Effected by the Use of Red Charcoal.....	VI	204	1878
Experiments with Straight or No-Bosh Blast-Furnace.....	XIII	498	1884
An Improved Langen Charger.....	XIII	528	1884
Comparison of Some Southern Cokes and Iron-Ores.....	XV	754	1887
Development of American Blast-Furnaces.....	XIX	992	1890
The Iron-Mining Industry of New Jersey.....	XX	224	1891
American Blast-Furnace Practice.....	XX	266	1891
Manganese in Cast-Iron.....	XX	315	1891
The Magnetic Concentration of Iron-Ore.....	XX	595	1891
Preservation of Hearth and Bosh-Walls.....	XXI	118	1892
The Magnetic Iron-Ores of Ashe County, N. C.....	XXI	277	1892
Discussion on the Crushing of Iron Ore.....	XXI	548	1892
The Hugh Kennedy Hot-Blast Stove.....	XXI	735	1892
Nickel and Nickel-Steel.....	XXV	961	1895
Removal of Sand from Waste-Water of Ore-Washers.....	XXVIII	842	1898
Important Results Obtained in the Past Fifteen Years with the Stiff and Heavy Rail-Sections.....	XXIX	1015	1899
Blast-Furnace Practice.....	XXXVI	796	1905

Mr. Birkinbine always maintained a friendly interest in his fellow members of the profession, and held to the thought that engineers were co-operators and not competitors. He continued his personal interest in all associates and was ever ready to help young men by advice.

As a citizen he was always active in promoting the public good. He served on the Civil Service and other Commissions, as well as rendering professional services for the City of Philadelphia. His neighborly activities were maintained up to the end of his life. His last days were happily spent among his family, for as a devoted and Christian husband and father he fulfilled what he considered his greatest pleasure and the noblest of all his duties.

He died May 14, 1915, at his home in Cynwyd, near Philadelphia. Simple funeral services and private burial were held in accordance with his known desire; but the former were attended by representatives of many organizations and by professional colleagues and personal friends from many places.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant

BY W. P. LASS, *B. S., JUNEAU, ALASKA

(San Francisco Meeting, September, 1915)

THE gold precipitate from the zinc-dust presses in the cyanide plant of the Alaska-Treadwell Gold Mining Co., Treadwell, Alaska, is treated, in the refinery adjoining, by the Tavener or lead-smelting method. About 3 or 4 tons of this precipitate are produced monthly having a gold assay value of \$40,000 to \$60,000 per ton.

It was formerly the practice to treat the by-products from this process, consisting of 2 tons of slag, 300 lb. of matte, 500 lb. of refinery refuse, flue dust, etc., in a 24-in. water-jacketed blast furnace. This practice was discontinued in the summer of 1914 by reason of the difficulty of keeping the lead-well open when treating a high-grade lead product; of preventing the loss of gold in the flue dust; and of avoiding injury to the general health of the refinery operators; a plain single-phase electric furnace was substituted for the blast furnace.

The furnace was constructed from an old steel acid drum by cutting off the top and introducing a cable, made from strands of bare copper wire, through the bottom and spreading the strands out fan-shaped on the inside of the drum.

Powdered graphite, obtained by grinding up old crucibles, mixed with 10 per cent. cement, was tamped wet into the bottom of the drum, around and completely covering the copper wires. The graphite was carried up to the bottom of the furnace, or lead-well, and acted as the lower electrode. The sides were built up of ordinary firebrick forming a melting chamber 14 in. in diameter by 20 in. high.

The upper electrodes, of graphite or carbon, are 3 in. in diameter and 40 in. long, arranged with joints enabling new electrodes to be connected without shutting down or wasting stubs. A screw feed was arranged for raising and lowering the upper electrode. (Figs. 1 and 2.)

The cover for the furnace had three openings, one for feeding the charge, one for the escape of gases, and one in the center for the introduction of the electrode. It was later found more practical to en-

* Formerly Cyanide Superintendent, Alaska-Treadwell Gold Mining Co.

large this center opening to 6 in. in diameter, to allow of a central feeding of the charge around the electrode.



FIG. 1.—ELECTRIC FURNACE FOR GOLD REFINING.

A 4-in. pipe connected to a ventilating fan carries off the escaping gases.

The furnace is operated on the lighting circuit through a 50-kw. transformer, 60 cycles, 110 volts.

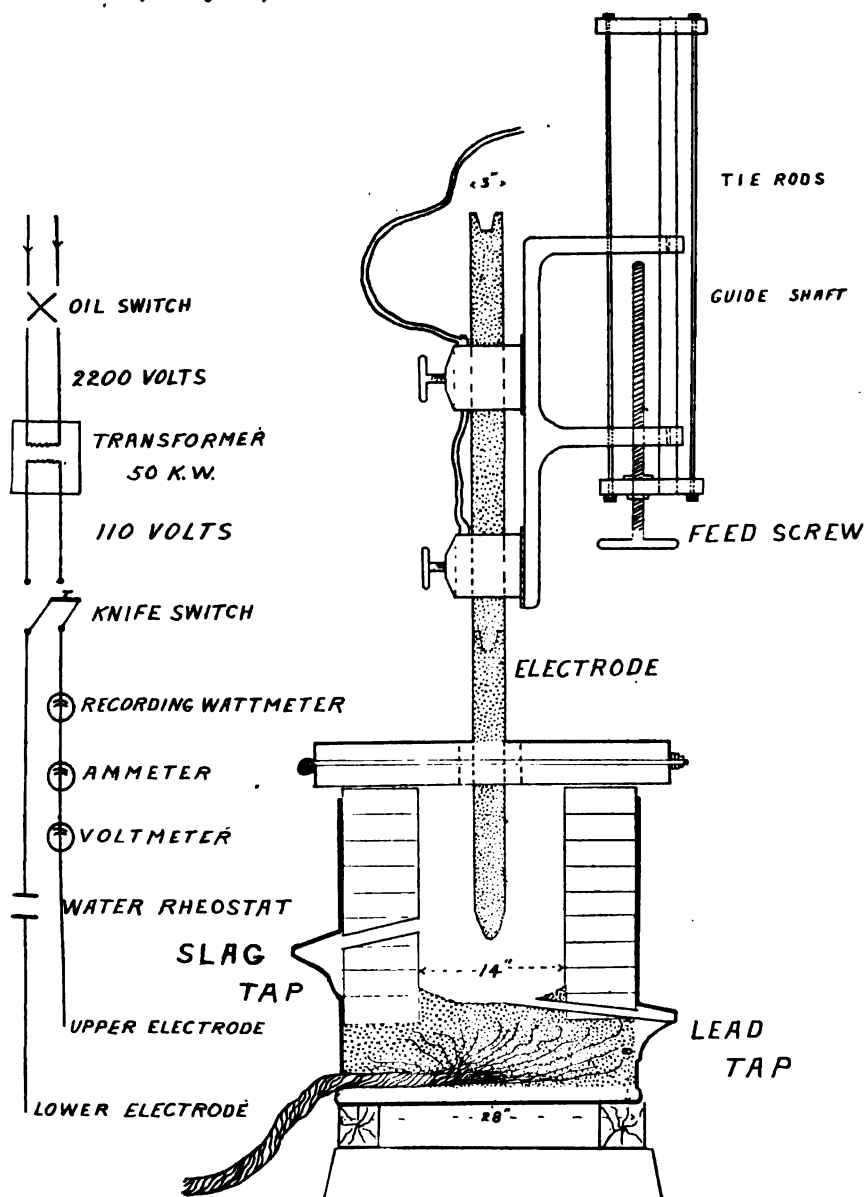


FIG. 2.—SECTIONAL VIEW OF ELECTRIC FURNACE.

Later, a water rheostat, operated by raising and lowering one disk on another submerged in a barrel of water, was constructed to lower the voltage for use when starting the furnace, or when there was a bath

of metal in the bottom of the furnace that would otherwise allow too much current to pass and thereby cause short circuiting.

The material to be treated in the furnace is mixed in the following proportions: Refinery by-products, 100 lb.; old reverberatory hearths, containing 60 per cent. CaO , 20 lb.; litharge, 20 lb.; coke, 2 lb.; scrap iron, 3 lb. About 160 lb. of the mixture is added for a charge. The total amount of material added during the melt is 8,507 lb., of which 5,870 lb. is by-product material (old slag, the residue from burning old mill launders, sweepings, etc.) of the following composition:

	Per Cent.
SiO_2	40.1
Fe.....	16.0
CaO	16.1
Cu.....	5.9
Pb.....	3.0
Zn.....	5.7
Al_2O_3	2.0
S.....	3.6
Moisture.....	4.0
	96.4
Gold.....	\$1,095.50 per ton.

Coke is added to the charge as a reducing agent, only a quantity sufficient to throw down the lead being used.

The charge is introduced into the top of the furnace through the central opening around the electrode without removing the slab cover.

In starting, the furnace is operated as an arc furnace, until it has become thoroughly hot, when the charge, consisting of slag, brick dust, etc., is added, the electrodes being raised as the load in amperes increases, until the entire chamber is filled with the charge and the upper electrode extends into the melt almost a foot. As the charge continues to melt the electrode is moved down or up, keeping the amperage reading as nearly constant as possible. The raising or lowering of the upper electrode decreases or increases the power consumption and thereby affects the amount of heat generated.

At this stage the furnace is no longer operating as a simple arc furnace, but has automatically transformed itself into a resistance type of furnace, the semi-fluid or molten charge acting as the resistant, the current passing from one electrode through the center of the charge to the other.

After the charge is in quiet fusion, which takes on an average 2 hr. 10 min., the power is turned off and the charge allowed to settle for 15 min. before tapping the slag. This allows the lead to settle out, the furnace acting as a forehearth.

In operation, the molten lead and slag are tapped, as in the blast

furnace. If the lead freezes it is melted by diverting current from the upper electrode and letting it pass through the lead, the tapping rod playing as an arc on the side of the furnace or into the lead tap.

The operators use colored or amber glasses to protect their eyes from the glare of the electric arc.

Operating Data

Total running time.....	128 hr. 40 min.
Average weight of charge added, pounds	160.5
Average fusion time of charge.....	2 hr. 25 min.
Total number of charges.....	53
Graphite electrodes used.....	1 per 24 hr.
Total power for 128 hr. 40 min., kilowatt-hours.....	4,440
Power used per hour, kilowatts.....	34.50
Power used per pound of material fed, kilowatt-hours...	0.52
Power used per ton of material fed, kilowatt-hours.....	1,044.0
Acheson graphite used per hour, feet.....	0.15
Material melted per hour, pounds.....	67.7
Material melted for 24 hr., pounds.....	1,625.0

The only item of cost for flux is the coke, 2 lb. per 100 lb. of by-product material or 118 lb. total being used, since the old reverberatory hearths furnish the lime, the cupels from bullion refining the litharge, and mill scrap the iron.

Both graphite and carbon electrodes are used, the former costing \$2.95 each, and the latter \$1.20. Power cost is less than 1c. per kilowatt-hour.

The labor required for the operation of the furnace, including the work of charging and tapping, is one-half one man's time, when melting 1 ton in 24 hr.

The furnace was constructed and put into operation by the regular cyanide-plant crew, without the aid of special electricians. The operation is so simple as to require no special training. Although the operators were unaccustomed to handling electrical equipment, no trouble has been experienced from electrical shocks, since the bottom of the furnace acts as the lower electrode, or grounded circuit.

The advantages of the electric furnace compared to the blast furnace for melting high-grade gold slags are: A saving in mechanical loss of gold in flue dust, because the melting is done in a quiet neutral atmosphere, instead of in a rising blast of air; the obtaining of a lower-grade slag, free from shot, by reason of the quieter melting action and the higher temperature obtainable, making a more fluid slag; the nicety of regulation of the melting temperature; the benefit to the general health of the operators.

The Rôle and Fate of the Connate Water in Oil and Gas Sands*

Continued discussion of the paper of ROSWELL H. JOHNSON, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 221 to 226. See also *Bulletin* No. 101, May, 1915, pp. 1157 to 1162.

E. W. SHAW, Washington, D. C. (communication to the Secretary†).
—With the word petrology—literally the science of rocks—being used primarily for igneous rocks, and with geophysics coming to mean the study of lavas and the rocks into which they solidify, contributions to the knowledge of the physical chemistry of sedimentary rocks, even though only ideas which may be used as working hypotheses, become most welcome. The forces which control the formation and subsequent history of the stratified portions of the earth's crust are so little known that no one has yet been able to settle finally the questions as to whether petroleum is of organic or inorganic origin; whether it originated far from its present position, or nearby. Yet both the chemical and physical problems of the rocks in which oil occurs seem to be more simple than those of the igneous rocks. The oil-bearing rocks of western Pennsylvania, for example, have in all probability never been subjected to a pressure exceeding 3,000 or 4,000 lb. to the square inch, nor to a temperature higher than 100° or 200° C. These pressures are considerably below the crushing strength of rocks, and the temperature far below their melting point. The igneous rocks, on the other hand, have been affected by enormous pressures and very high temperatures and yet steady progress is being made in reproducing in the laboratory the conditions of their formation and subsequent modification within the earth. The solution of the sedimentary-rock problems would seem to be fully as important as those of igneous rocks, and yet we must content ourselves with reasoning backward and our inferences as to what has happened are oftentimes little more than guesses.¹

One of the fundamental problems is the origin of the salt water in oil fields. Is the salt water fossil sea water; has it been formed by the solution of salts of various kinds from distinct beds, from disseminated particles, and from the weathering of suitable grains in the rock; or is it of deep-seated origin? Its general resemblance to sea water points to the first, but the principles held by some concerning the free circulation of water throughout the earth's crust, and certain peculiarities of composition indicated by the few analyses available, seem out of harmony with such an explanation.

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†Received Apr. 16, 1915.

¹The need of work on sedimentary rocks is indicated by a question put to the writer by an eminent authority on economic geology, to this effect: "Is salt water at all common in oil fields?"

We must grant that the strata before becoming deeply buried, if not at the exact time of their formation, were saturated with water. Those containing the remains of marine organisms or interstratified with such beds were, except for gas bubbles, filled with sea water. Many layers of sand which were once parts of sand dunes, and also many layers of sand deposited in fresh water, have later been submerged beneath the sea or lowered to such a position that salt water flowed into them. Some beds of fresh-water origin have never been below sea level, and hence may never have contained any salt water, but such beds form only a very small fraction of those in any way associated with oil and gas.

It is also incontrovertible that the sedimentary strata are now more compact than they once were, and that part of the water with which their pores were filled at or shortly after their formation, has been forced out; also that shale has suffered much more compacting than sandstone. It seems reasonable to assume, further, that the main part of the compacting took place in each stratum within a comparatively short time after its formation, and that probably the time of greatest settling was shorter for sands than for clays, and probably shorter for clays than for coal. But beyond this, inferences concerning the history of connate water are little more than assumptions. Has it remained in the rocks since their formation? It is unthinkable that each pore has retained its own water from the start. Then how far has it traveled: to other parts of the same bed, to other beds in the region, or to other regions?

In the first paragraph of his paper Professor Johnson uses the expression, "The deeper sands carry increasingly less proportionate amounts of water." This conclusion has been frequently stated, and seems to have gone unchallenged, but the writer has searched in vain for proof that the deeper sands actually do contain less water. The idea seems to be held that certain deep-lying sands have empty pores, as is shown by the fact that they absorb water instead of supplying it. But cannot all the phenomena be explained by the factors which control the flow of water in deeply buried porous strata? Is it reasonable to suppose that the pores of any sand have nothing in them? If they contain water, oil, natural gas, or air, they cannot absorb more liquid or gas unless some of that which they contain is either given up or compressed into a smaller space. In general, the deeper the sand below the surface, the more likely it is to be effectually sealed in; hence it is conceivable that a deep sand saturated with water may be so completely surrounded with shale that no air or other gas or liquid can enter it to take the place of the water which would otherwise flow out into a well. Furthermore, it is also conceivable that a sand at any depth connected with the surface at, for example, two widely separated points, may have water flowing through it under such a slight head that the weight of the water in a well which is sunk to the sand may cause it to flow from the well into the sand. The sand may thus have its

pores completely filled and yet actually absorb water. If the known facts allow of such an interpretation, the reports of deep-lying dry sands, or even of less water in deep sands, should not be taken as final.

In the second paragraph, the belief is expressed that oil and gas have originated in shale near their present location. The field evidence seems to the writer to admit of quite a different interpretation, though neither interpretation is sufficiently well supported to be more than an idea or working hypothesis. The writer has found that recent mud deposits along the sea coast have a weight per cubic foot ranging from a little more than that of water to about twice that of water. The total amount of solid matter per cubic foot in some samples weighs less than one-tenth as much as a cubic foot of the same matter without pores of water would weigh. The water content of various muds tested ranges from 40 or 50 per cent. to 90 per cent. or more. In the compacting of such muds into shale, a large part of this water is expelled, and if oil and gas originate in such muds, the possible distance to which they may be carried by such outflowing water seems to be great, for not only a part of the water originally contained in a particular part of a bed moves through it, but also water from other beds and other parts of the same bed.

The idea that sand spits become buried and some of them form oil reservoirs is commonly entertained, but seems to the present writer quite untenable. The material of sand spits, beaches, etc., is rarely buried without being completely reworked, and changed into some other form of deposit. In an advancing sea beach deposits are formed at the shore and gradually pushed inland as the sea advances. They are continually subjected to a reworking process, and such parts as are not carried landward by the advancing sea are spread out in the form of bottom deposits.

That organic muds lie at various positions below the surface, and in places are continuous to great depths, seems too well established to need argument. At the mouth of the Mississippi, for example, carbonaceous muds from which great volumes of marsh gas are given forth are found at all depths down to nearly 2,000 ft., the depth of the deepest well yet bored. The argument of Craig referred to seems somewhat misleading, but not altogether incorrect. Craig says that "The fallacy lies in the assumption that these samples from the upper layers of sludge (in harbors, mud flats, etc.) are typical in chemical composition of the mass of slowly accumulating material beneath." Studies made by the writer indicate that more organic material is present at the bottom than a few feet below, but it is nevertheless found at all depths. Craig argues that all or nearly all animal matter oxidizes before it is buried, and the writer knows of no evidence in recent deposits which is incompatible with this view. The buried carbonaceous matter may be almost entirely vegetable, but at the same time, Professor Johnson's belief that organic matter is present at considerable depth is well supported.

The idea is conveyed that the water which is being pressed out of layers of mud and sand takes a course which is dominantly upward. But oil-field strata are commonly nearly horizontal, and hence, water which is being pressed out of mud and sand should move much more laterally than vertically. Also, in order to have an outlet at the surface, the sand need not be a single persistent bed extending to some point sufficiently low to allow of flow in that direction, but it may have zigzag connection, through interfingering lenses of sand, with the surface at some more or less distant point.

The possibilities of gravitational sorting depend upon several factors. To some investigators of petroleum it has seemed quite possible that without any lateral movement, oil, water, and gas may take positions with reference to their specific gravities within a layer of sand, the particles of one fluid moving past another in the minute pores of the sand. A few years ago the writer obtained some glass tubes of various diameters, up to about $\frac{1}{4}$ in., and 6 ft. long, and filled them with alternating globules of oil, gas, and water, in order to determine how large a pore is necessary for gravitational sorting. M. J. Munn, whose hydraulic theory was such an important contribution, was associated in the experiment. It was found that only in the larger tubes was there any perceptible movement. Some would probably remark concerning such experiments, that what would not happen in the laboratory might happen in nature, because of the millions of years available. This suggestion seems inapplicable, because in a tube more than 50 times the diameter of the average space between sand grains in oil reservoirs no movement took place in several weeks. The frictional resistance to movement increases so rapidly with decrease in size of pores, that it seems beyond possibility that oil and water could change places under the influence of gravity in such small spaces as are available in ordinary sands.

The only possibilities of gravitational sorting without other movement appear to be the following: First, in a coarse sand with very fine globules of oil in water—almost an emulsion—such a movement may take place; second, if through surface tension, slight movement, or some other factors, oil comes to fill the pores in a considerable body of sand in the lower part of a stratum elsewhere saturated with water, it seems possible that the body of oil as a whole might be floated up to the top of the sand and water take its place below, without any lateral movement. If, however, movement can be postulated, gravitational sorting may take place very readily. To illustrate, if a pore between sand grains contains both oil and water, and there are two openings leading to adjoining interstices, it is easy to imagine that the oil would take the upper and the water the lower opening, other conditions, of course, being well balanced.

The expression "capillary attraction" seems to be frequently used in literature on petroleum without an adequate realization of its controlling

factors. As a result, conclusions are sometimes drawn and statements made which are open to question. For example, in Washburne's important contribution on Capillary Concentration,² it seems to be assumed that capillary attraction is independent of the material forming the sides of the capillary opening. It is stated that "liquids are drawn into these pores by the force of capillary action, which varies directly as the surface tension of the liquid and inversely as the diameter of the pore. Crude oils probably have only about one-third the surface tension of water, and hence only about one-third of the capillary power." Now, the fact is that capillarity depends not only on the surface tension of the liquid but on the nature of the surrounding substance. It is, hence, unsafe to conclude from determinations of the height to which water rises in a glass tube that a similar force will be exerted upon water by clay or some other substance. A glass tube in which water will rise to a height of an inch or a foot will, if greased, have quite a different effect upon the water. Indeed, its pull may be negative, so that the water in the tube may stand lower than in the surrounding vessel. It must be remembered that capillarity requires three substances, water, glass, and air, for example, and that at the contact between shale and sandstone saturated with water, no capillary attraction will be displayed.

As to the relative viscosity of oil and water, it seems probable that the difference, particularly in deeply buried sands, is not important, because the temperature is higher and the viscosity lower than at the surface. The difference in viscosity between warm oil and cold oil is greater than that between average oil and water at ordinary temperatures.

The statement to the effect that oil does not readily penetrate pores whose walls are wet with water seems at first thought reasonable. We oil our boots and our roads partly to make them shed water. But it may be remarked that both water and oil are attracted by the clay and sand walls of pores, and the frictional resistance to movement of the fluids thus gripped is great, at least for the molecules adjacent to the walls. On the other hand, water does not show such an attraction for oil, and hence a minute globule of oil may pass quite readily through a pore having a larger diameter lined with water. Globules of oil inclosed in water in a glass tube rise more readily to the top than oil globules which come into contact with the glass. However, one may reasonably doubt the existence of oil globules sufficiently small to pass through the finer pores without touching the walls.

The amount of oil which may be carried by gas bubbles as films lining the bubbles is apparently small, for the oil films are little if any thicker than molecules and a very large amount of gas would be required for the transportation of a minute quantity of oil.

² *Bulletin* No. 93, September, 1914, pp. 2365 to 2378.

Professor Johnson accepts the idea that some oil and gas are produced by pressure. From the standpoint of physical chemistry this seems somewhat doubtful, and poorly supported by laboratory results. The few experiments that have been performed seem to indicate that when carbonaceous deposits are subjected to great pressure in the laboratory, they are not metamorphosed, and no oil, and commonly no gas, is produced. If water is present and allowed to escape the sediment is compacted, but if the water is not allowed to escape the sediment is unchanged. The conclusion that oil may be produced by dynamochemical agencies appears to rest upon rather inconclusive geological evidence, and to be unsupported by general principles. On the other hand, if the temperature is raised, it is not at all difficult to obtain various hydrocarbons from any carbonaceous deposit. A little more geophysical investigation of this subject is greatly needed.

The inference that some oil is biochemical is also in need of better support. If it is correct, oil films should be common at the surface of recent deposits, but they are very scarce. In fact, the writer has not found them except where some animal remains were decomposing. Bacteria do not thrive far below the surface. They have been reported at 20 ft., but this report is doubted by some. The writer has sunk many holes in marshes, to depths of 50 to 100 ft., but has found no sign of oil.

The idea that the character of coal and oil of western Pennsylvania shows a relation to the horizontal thrusts which have produced the folding in the Appalachian Mountains—that the thrust pressures have affected the rocks west of the Alleghany front in a gradually decreasing amount, seems to the writer rather improbable. About the only indication of thrust in western Pennsylvania and Ohio is the fact that the anticlines and synclines have a general trend more or less nearly parallel to the Appalachians, but can such low and multitudinous folds be produced directly by lateral thrust? Where beds first begin to yield to lateral thrust they are thereby weakened and there is a tendency for the shortening to be effected by a single high fold, though this tendency is modified somewhat by heavy loading. The shortening involved in the low folding of the oil and gas region is so slight, and the wrinkles are so generally developed, that it seems much more probable that they were produced by forces acting principally from below, among which gravity is dominant. An anticline 30 ft. high and 3 miles broad means a shortening of only a tenth of a foot. According to Bailey Willis, in the eastern part of the Appalachian region the principal component of the thrust was horizontal, whereas in the western part it was vertical. If the quality of coal and oil west of the area of pronounced folds shows a relation to the Appalachian Mountains, may it not be due to greater loading on the east? Some of the formations are known to thicken in that direction, and there is good evi-

dence that more material has been removed by erosion from eastern Pennsylvania than from the western part of the State.

One of the most interesting parts of Professor Johnson's paper is the paragraph showing that he regards it possible that some gas is of inorganic origin. The close confinement of most gas pools makes the possibility of accession of deep-seated gas seem remote. The fact that gas will be held under a pressure of hundreds of pounds to the square inch for many years by a stratum of shale a very few feet in thickness, makes it difficult to believe that any gas has traveled upward through many thousand feet of very compact rock; yet the state of knowledge of sedimentary rocks is such that the conclusion that such a migration is impossible would be altogether unwarranted. Besides, there is the possibility of transfer by solution, for water will dissolve hydrocarbons to some extent.

The conclusion is drawn that the cement in sandstone comes principally from below. Is it not quite possible, however, that a large part of the cement came from within a small fraction of an inch of where it is now found? Granular materials under pressure, containing water under less pressure, tend to dissolve at the points where the grains are in contact, and the dissolved material is commonly redeposited very near the point of solution. If the water and rock are affected by the same pressure, solution and redeposition will not take place, and this may explain the varying degrees of cementation displayed by sandstones of apparently similar history. Even where the cement is not of the same material as the sand grains, is it not possible that it has come from various directions, according to the movements of the water?

In connection with the idea that temperature rises in great masses of sediments on account of rise of isogeotherms—that these masses have sunk into the earth to positions where the temperature is higher, it should be remembered also that on account of deformation and erosion, the temperature throughout the mass of strata has been affected at different times in different directions. It has seemed to the writer that these changes may play a part in the accumulation of oil and gas and water, because any little motion, if it amounts to a distance of only a few times the diameter of a pore, and especially if oscillatory, aids in the segregation of oil and gas.

The inference that, other things equal, the greater the pressure the greater the amount of gas, and also that high pressures tend to produce gas rather than oil, is, if true, of great importance, but at present it seems to be in need of more supporting data.

The point is made that tortuosity and fineness of pores affect the head of water. Is hydrostatic equilibrium affected by such factors? Where a fluid is moving through pores in a rock hydraulic forces are involved, and the effect on the water head may be considerable, but where movement is wanting, the fluid should in time come to equilibrium and the head of the

water should be the same without regard to size or tortuosity of the pores unless gas or oil is mixed with the water so that capillarity or some other factor enters the problem.

The height to which water will rise in a well may be affected also by atmospheric pressure if the sand is an effectively sealed-in lens. The areas of the upper and lower surfaces of a sand layer are generally so vast, compared with the areas exposed in the sides of a well, that as a general rule there is, somewhere, a few acres where water may slowly enter the sand over an area so much larger than the walls of the well that it may take the place of a considerable volume which flows out into the well. Since the greater the depth the greater the possibility of sealed-in sands, the lower sands should in general yield less water and be reported drier.

The head of water in a sand evidently depends on several factors and has a wide range. In a sand outcropping down the dip but not up the dip, it may be zero or even a minus quantity, so that water will flow from the well into the sand. The maximum height to which pressure may rise is even more problematical. Obviously it may range up to the weight of a column of water, or nearly 450 lb. per square inch for each thousand feet of depth. In lenses which become effectually sealed in before settling ceases, the pressure may become as great as the weight of the superincumbent load of rock, or about 1,200 lb. for each thousand feet of depth. If in such a sealed-in lens some chemical change takes place, such as the development of gas or precipitation of cement, it seems conceivable that the pressure may, through the tensile strength of the rock, rise even higher without producing an upheaval, just as the pressure in a boiler may be greater than the weight of the steel forming the upper part of the boiler. If such pressure were developed over a very large area the coherence of the rock would of course become of small importance, but oil and gas pools are not so large, and the pressure does not rise so high as to make the coherence of the hundreds of feet of strata negligible.

The conclusion that "We find, then, that the oil and gas reservoirs, when newly compacted, contain a large percentage of the oil which they are to contain," does not seem to be in harmony with the character of recent deposits. The Mississippi delta, for example, contains a great abundance of organic material of vegetable, and perhaps some of animal, origin, and from this material immense quantities of marsh gas develop, but no trace of oil has been found, though several wells have been sunk to depths exceeding 1,000 ft. Further, no hydrocarbon gas, other than methane or marsh gas, has been found.

The lateral movement of water and oil is thought to be slight (p. 233). But in upward movement the fluid must traverse clay and shale in which such movement is opposed by not only hydrostatic head but great friction. On the other hand, in a direction more nearly horizontal the fluid may flow through sandstone where the hydrostatic head is nearly the same and the

frictional resistance to slow movement is probably much less, though the distance is greater. The facts that oil and gas generally occur in great basins, that sands may either extend out to the surface or have a connection somewhere with some other sand which outcrops toward the margin of the basin, and that except in hard fractured strata it is easier for fluids to move along beds than across them, make it seem probable that the fluids in being forced upward move much more laterally than vertically.

Concerning the statement that "gas is not to be expected in large quantities and under high pressure" in recent deposits, it should be remarked that immense quantities of marsh gas are included in recent deposits, though ordinarily not well segregated, so that, on account of dissemination and the difficulty of recovering gas from soft materials, few attempts are made to collect and make use of it. May not the greater amount of gas recoverable from deposits which have become somewhat consolidated be more reasonably ascribed to the fact that there has been sufficient time for accumulation in well-defined pools and the clays have become sufficiently tight to retain the gas, rather than to a greater production of gas at a later stage on account of greater pressure?

Would not the third conclusion (p. 224) be more correctly stated if the wording in the latter part were "make available a larger amount of oil" instead of "a larger proportion of the oil which was originally formed?" The proportion of available oil to organic material might, it would seem, be fully as great in an area of thin shale and thick sandstone as in an area of thick shale and thin sandstone, particularly since sandstones are sufficiently variable in porosity to affect the location of pools. Whether Johnson's conclusions or those of the other investigators referred to are correct, it may be contended that the oil in thick sands has a better chance to escape than that in thin sandstones.

Professor Johnson evidently regards as a possibility, the idea that a considerable amount of oil may have come from limestones (p. 224). This inference seems to be opposed by certain facts, one of which is that calcareous and carbonaceous materials are rarely deposited together. In peat bogs shells are generally absent, and in seas where lime is being deposited there is generally complete decay of the organic matter. Beds of limestone, coal, and carbonaceous shale are commonly interstratified, but the materials are generally not mingled in a single layer. The character of the very recent lime deposits in Florida, for example, is such as to make one dubious concerning the conclusion that a globule of oil in a Paleozoic coral is the remainder of a part of the body of the coral. It would seem much more likely that the globule of oil had migrated from some other place and stopped in the coral because of the cavity. How far the globule may have traveled one may scarcely guess. Some phenomena displayed by oil suggest that, given geologic time, it may journey for very long distances. It has been spoken of as a "wandering Jew"

the possible extent of whose travels is almost unlimited. On the other hand, the effective inclosure of oil and gas pools and the high pressures developed tend to make one assume that the oil and gas have not migrated far. Perhaps, however, the effective sealing-in of pools may take place long after the deformation of the stratum, and may be preceded by a long time, during which migration is easy.

Concerning the depth and conditions under which bacteria may live and operate, knowledge is scant. Anaerobic bacteria have been reported at a depth of 20 ft. in bogs, but whether they are sufficiently common at such depths to be important factors, to say nothing of whether they may effect important changes at greater depths, remains to be determined.

The conclusion that gas is more abundant than oil at great depths requires that pressure is more effective in producing gas than oil, and that neither gas nor oil travels far. The idea that the oil is unavailable in deeply buried strata because it has been forced into the shale, seems out of accord with the nature of strata which have been deeply buried. There is much pore space in Cambrian sedimentary rocks, and therefore for oil to be forced out of them requires that some of their pores be filled with gas and that no openings out of the surface exist. On the contrary, Cambrian sandstones are commonly loose and porous, and the writer would expect that even if they were at first filled with oil, the pressure would push the oil diagonally along the bed toward the outcrop before it rose high enough to force the oil into shale. If the sand were not saturated with oil at the start, the remaining pore space, except near the outcrop above the surface of ground water, would be occupied with water which would all be forced out, either into shales or at the outcrop, before any oil would enter shale.

The statement is made that "relatively little of the deeper waters reach the surface, owing to the interposed barriers." If this were true the waters of the deeply buried sands should be under greater pressure than that exerted by the superincumbent load of rock, and when drilling deep sands containing water only, we should have a series of water gushers each followed by a certain amount of settling. On the contrary, it seems to the writer probable that the water in all sands is gradually forced diagonally upward to the surface through settling of strata and filling of pores, and that, although great resistance is sometimes encountered, this resistance, except in sealed-in lenses, rarely amounts to more than that of a hydrostatic head of water, filling the overlying rocks.

Is it true that the opinion is generally held that oil is "lost at an outcrop, because water displaces it by gravity"? The ideas that some seeps are produced by oil being forced out by pressure from below, or that some are due to hydraulic pressure exerted through the stratum from which they arise, seem tenable. Whether in the Mexican fields the oil adjacent to intrusives has been carried up by water squeezed out of the strata below, or has been carried by the intruded mass, or has not been

carried up at all, but has formed in approximately its present position, are unsettled questions.

The conclusions at the close of the paper are based upon the preceding argument, and insofar as it is uncertain, the conclusions are doubtful. The second conclusion, in particular, seems to the writer to need modification, because it involves "absence of water" in deep sands. This inference, as stated before, though generally regarded as well established, seems very unlikely on theoretical grounds, and not proved by field evidence.

Professor Johnson's inferences are evidently the result of much careful thought and although the present writer feels inclined to a different opinion concerning many of them, few can now be actually disproved. In any case, the paper is full of ideas and stimulating to thought.

The Mayari Iron-Oré Deposits, Cuba

BY JAMES F. KEMP, NEW YORK, N. Y.

Postscript to a paper presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 129 to 154.

The writer looked over the quarry with care in the search for fossils, having in mind shells or corals of fairly large size. In the brilliant summer sunshine, none were observed on the glaring white ledges. The impression was gained that none were present and the statement to this effect was made in the edition of this paper published in the *Bulletin* of the Institute for February, 1915, p. 136. On closer examination in the laboratory it has been discovered that the limestone itself is made up of minute organisms, which are clearly revealed by microscopic examination of thin sections. The principal component of the rock is a small foraminifer, 0.1 to 0.3 in. (2.5 to 7 mm.), of the genus *Orbitoides*, presumably an Oligocenic form. Figs. 1 and 2 illustrate it. The accompanying description has been kindly summarized by Marjorie O'Connell, M. A., Curator in Paleontology in Columbia University, who first detected the evidences of organisms in the rock.

Orbitoides kemp O'CONNELL N. SP.¹

The foraminiferan which makes up the Seboruco limestone is an *Orbitoides* of a species new to North America. The only *Orbitoides* heretofore described from this country is *O. mantelli* from the lower Oligocenic of the Gulf States. This form from Cuba, then, so different in all of its characteristics from the *O. mantelli*, is of great interest as perhaps forming a basis of correlation over wide areas. The most closely related species is *O. dispana* Sowerby, which occurs in the Nummulite limestone of the Eocene of the Bavarian Alps, in the Priabona beds of the lower Oligocenic of Massano, and elsewhere in Italy, as described by Gümbel.² The age of the limestone in Cuba would thus seem to be of upper Eocene or lower Oligocenic age, probably the latter.

The fossil has the form of a sphere surrounded by a wavy rim of horizontal extent, the whole resembling two flat-brimmed, round-crowned hats placed base to base. The total extent of the shell ranges up to 7+ mm., while the thickness of the central sphere is 2.5 mm. in one of the largest individuals. Like all of the other species of *Orbitoides*, this form begins with a single layer of chambers which multiply in one plane and assume a circular outline; but it is in the subsequent growth that the distinguishing specific characters appear, for the chambers at the central portion of the shell increase more rapidly than do those near the periphery and in consequence a swelling is produced which finally gives the spherical form which is so characteristic. The rim, on the other hand, shows little chamber-increase and remains thin. This is clearly shown in the sections figured, where the central row of larger chambers forms the equatorial zone. Radiating from this row, in section, there appear a number of solid cones alternating with broad reticulated ones. The solid cones mark the more dense structure forming the walls surrounding the reticulated or chambered cones. These appear upon the surface as distinct pits, whereas the solid part appears as raised walls.

¹ Named in honor of the discoverer of the fossil, Prof. James F. Kemp of Columbia University. A complete description of this fossil will be published elsewhere.

² C. W. Gümbel: Beiträge zur Foraminiferenfauna der Nordalpinen Eocägebilde, *Abhandlungen der Math.-Phys. Classe der königlich-bayerischen Akademie der Wissenschaften*, vol. x, pt. 1, p. 701 (1866).



FIG. 1.—CROSS-SECTION OF *Orbitoides Kempfii*, IN LIMESTONE FROM THE QUARRIES SHOWN IN FIG. 3. ACTUAL FIELD, 2.5 MM. OR 0.1 IN.



FIG. 2.—SECTION SHOWING THE TAPERING EDGE OF *Orbitoides Kempfii*, AS WELL AS MINUTE FORMS OF OTHER ORGANISMS. ACTUAL FIELD, 2.5 MM. OR 0.1 IN.

Some Defects of the United States Mining Law

Continued discussion of the paper of COURTENAY DEKALB, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 331 to 337.

R. B. BRINSMADE, Puebla, Pue., Mexico (communication to the Secretary*).—I agree heartily with Mr. DeKalb's five reasons why the United States mining law should be revised. I desire to criticise only certain ideas expressed under the headings "Initiation of Mineral Rights," "Titles," and "Universality of the Law."

Whatever the "spirit and intent" of the mining law, I consider that 42 years of practice have shown that it subserves the interests of neither the poor prospector nor the average citizen.

Considering the prospector, it is clear that his fundamental need is not freedom from taxation, but *good* mineral ground *available* for prospecting. Even under the equitable laws of primitive placer mining, when each worker is only allowed to locate the gravel which he can personally handle, the first comers get the choice of locations and the late arrivals either get inferior claims or find none at all unoccupied. But when each first comer is allowed to locate a large area, merely for speculative or forestalling purposes, as Mr. DeKalb acknowledges is now possible under the existing mining law, the handicap of late arrivals becomes so excessive as to discourage their productive efforts altogether.

A poor prospector would far rather pay the areal tax demanded by Peru (\$7.50 gold for metallic and \$3.75 for non-metallic ground, yearly, per hectare) and get a chance to work on *likely* ground, than to be locked out from everything but *barren* or *remote* locations by the patented or fraudulently held claims of the United States. A rich operator can often stand the expense of buying out a forestaller or of boring for deeply buried orebodies, but the poor prospector must have cheap ground and easily found mineral or give up the ghost.

Mr. DeKalb first eulogizes mining activity, and then proceeds to eulogize the freehold title because it permits a claimholder to take advantage of dilatory courts and to hang on to his claim long after he should have lost it for failure to pay his State taxes. My observations in many countries have convinced me that where such delays favor one worthy, but unfortunate, prospector, they enable lawyers, real-estate operators, bartenders and similar "mining men" to exclude a score of prospectors from all the likely ground in a whole camp.

I cannot understand how Mr. DeKalb can have "profound admiration and respect" for a mining law whose "tenure by assessment work is open to grave abuses" and "has unfortunately lent itself in this par-

* Received May 7, 1915.

ticular to the retardation of industry, to the purposes of the 'dog in the manger,' and to those of the grafter." But, even supposing that these abuses can be corrected (as Mr. DeKalb hopes) for *unpatented* claims, the granting of the patent or freehold title he recommends would at once relieve the land-owner from any further pressure to insure his activity, as, under the trivial land-value taxes prevailing in most States, he could then hold his ground idle indefinitely at slight expense.

Therefore it appears that the requisite activity of mineral claimholders can only be *perpetually* insured by refusal to grant them anything but the leasehold title. And this is the practice of Latin Europe and America, working under the Napoleonic Code. To *properly* enforce the assessment-work requirement on claimholders would require a complete reorganization of the Federal land office, both methods and personnel. As such a revolution seems remote, I believe mining activity could be more feasibly assured by substituting for the annual assessment work an areal tax of \$5 an acre for metallic and \$2.50 an acre for non-metallic minerals, or slightly greater than the successful areal tax of Peru, in order to conform with the higher wage-scale of the United States.

While the leasehold system, with a sufficient areal tax to prevent forestalling and monopoly, would satisfy all the needs of the poor prospector, it would still fall far short of doing justice to the average citizen. Mr. DeKalb, in his objection to the *direct* taxation of mineral land, evidently overlooks the *dual* nature of property which I have elsewhere explained.¹ As by the English common law² the ownership of even freehold land is retained by the nation—for "fee simple" means a feud or lease from the sovereign political power—it is evident that no government has a right to dispose of its mineral land to *certain* citizens without considering the rights of the *balance* to its usufruct.

If a man had a coal bed on his farm, he would not consider himself benefited by its development and extraction unless he also got a payment for the value of the coal in the ground. Similarly, no miners, however poor and worthy, should be allowed to extract and sell the minerals of a nation without paying to the government—the representative of the whole people—the royalty value of their output. A mineral-land policy should insure justice in the disposal of the landed heritage of the average citizen, even if thereby development be retarded or certain lucky individuals fail to get rich quick from the unearned increment in rich mineral deposits.

However, a direct and just tax on mines can be levied so as both to stimulate development and increase the average gains of mine operators. To achieve this, it is merely necessary to tax only the net profit or true

¹ Our National Resources and Our Federal Government, *Trans.*, xlv, 633 to 640 (1912).

² "Landed Estate," in Blackstone's *Principles of Law*.

mineral royalty, which is computed by subtracting from the gross proceeds the operating expenses plus the interest and amortization of the *true* capital invested (cost of construction and development). By this system a large number of mines would pay less taxes than at present and the tax quota of the industry would fall chiefly on those properties working bonanza orebodies. The details of this taxing system will be shortly published in my new book entitled *Landlordism*.

E. D. GARDNER, Missoula, Mont. (communication to the Secretary*). —In reference to Mr. Winchell's discussion in the May *Bulletin* of Courtenay DeKalb's paper on the revision of the mining laws: While agreeing with Mr. DeKalb and Mr. Winchell that there is a necessity for a revision of the mining laws, I wish to call attention to a misconception in Mr. Winchell's discussion of Mr. DeKalb's article regarding the present procedure concerning mining locations within the National Forests. Mr. Winchell states:

"A prospector is a man searching for mineral, for a vein, for a deposit. He wishes to find something to which he can later obtain title. He must necessarily be protected in his possession while he is searching for his vein, but under the present law he is a trespasser upon the public domain until he has found his vein. Twenty-five years ago it was perhaps an easy matter to make a discovery without any prospecting work. To-day it is exceedingly difficult. Twenty-five years ago there were no National Forests, there were no geological or mining experts appointed by the government roaming over the country and closely scrutinizing the prospective work of the attempted locator. To-day if a man makes a location within the limits of the National Forest, within a few days a geological expert comes along and says to him, 'What kind of a location have you? Let me see. If it is a lode claim, have you a vein in place? If it is a placer claim, have you rich enough material to pay for working?' If you have neither of these, then you have no location, 'skidoo.' A prospector must be assured of a reasonable possession of title while he is searching for his vein, as well as subsequently to its discovery. Under our present law he has no such right."

The government employs geological and mineral experts to make examinations of mining claims within the National Forests; but, unless the claims are actively interfering with the administration of the Forests, no examinations are undertaken until application for patent is made. The proportion of mining claims examined prior to application for patent is very small, probably less than 1 per cent. of the total number located. There are thousands of mineral locations within the National Forests of which no examination has been made by the Forest Service, and in all probability never will be except in cases where application for patent is made.

I discussed this matter in a paper entitled Mining Claims within the National Forests, presented at the Salt Lake meeting of the Institute.³

In many mining districts in the West there is some opposition to the

* Received June 7, 1915.

³ *Trans.*, xlix, 408 (1914).

practice of the government in having mining claims within the National Forests inspected before the issuance of patent. Many mining men feel that they are entitled to patent to mining claims, particularly in established mining districts, even if they have not made mineral discoveries on the individual claims. When such claims are contested they blame the Forest Service, while it is obvious that the grievance is against the mining laws.

Congress has set aside a part of the public domain for a specific purpose—for the growing of timber, and the protection of the watersheds. The fact that any ground has thus been withdrawn is *prima facie* evidence that it has a definite value for the uses prescribed by Congress. Is it unreasonable for the government to require mining claimants to comply with the Federal law before giving them absolute title to a part of its domain which has been designated by Congress as having a presumptive value for the purposes for which it has been reserved?

A prospector or miner is in no way disturbed by government agents in the enjoyment of his rights within National Forests. No examination is made of any ground located for mining purposes unless, as stated, it interferes with the administration of the Forest. A claim may interfere with the administration of the Forest when it conflicts with areas leased to individuals under what is known as a Special Use Permit; when it is included within a tract from which the government has sold, or contemplates selling, the timber; or where the claim is so located as to control rights of way over which it is necessary to transport Forest products.

At the present time, in some parts of the country, the timber business within the National Forests is seriously interfered with by unscrupulous locators of ground under the mining laws. In all such cases, the government has a direct interest at stake, and an examination of the interfering mining claims is made to determine their validity.

Are the Deformation Lines in Manganese Steel Twins or Slip Bands?

Addendum to the paper of HENRY M. HOWE and ARTHUR G. LEVY, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 99, March, 1915, pp. 587 to 600.

APRIL 14, 1915. An alternate explanation is that we have to do here with three sets of phenomena, slip bands, and twins of two orders of magnitude; that the deformation lines on the polished surface are slip bands, and the persistent lines found on repolishing and re-etching are twins formed along the slip planes. In addition to these twins, which on a repolished surface reproduce the general position which the slip bands have before repolishing, there are the broad twinned bands which give the staircase effect in Figs. 1 and 2.

In Section 5, p. 591, we gave as one reason for the industrial value of this alloy the ease with which it amorphizes. Since then we have found an illustration of this, by comparing the Brinell hardness as found on applying the load all at once in the usual way, with that which results from applying it extremely slowly,¹³ and we have applied this test to manganese steel, to the so-called "American ingot iron" with 0.01 per cent. of carbon, or nearly pure ferrite, and to eutectoid steel of 0.92 per cent. of carbon, or nearly pure pearlite.

The deformation starts a hardening process, which, as might be inferred from Muir's classical work,¹⁴ completes itself rather slowly. Hence retarding the Brinell test gives the step in this hardening process, started by each fraction of the deformation caused by the several successive increments of load, time to continue and thus to decrease the indentation which the later successive fractions of the load will cause, and thus in turn to increase the hardness number as found after the whole load has been applied.

The results, condensed here, show that manganese steel increases in hardness nearly 20 per cent. on thus retarding the testing, whereas pearlite, the component of any hypo-eutectoid steel with which manganese steel would have to compete for its special services, does not increase at all. On the other hand the "ingot iron" increased in hardness by no less

¹³ In the retarded test a load of 1,600 kg. was applied at 10 a.m. and increased by 200 kg. every 2 hr. till 6 p.m., when it had reached 2,400 kg. At 11 the next morning and every hour thereafter the load was increased by 100 kg., till 5 p.m., when it reached the usual 3,000 kg. It was then left on for the usual time, 30 to 60 seconds. This test was uninterrupted, in the sense that each fraction of the load remained on the specimen continuously from the time when it was first applied till the end of the test.

¹⁴ The Recovery of Iron from Overstrain, *Philosophical Transactions of the Royal Society*, vol. xciii, A, p. 1, and especially p. 13.

than 37 per cent. Even after this increase it remains of course extremely soft.

This failure of pearlite to gain hardness on retarding the Brinell test tallies with the observation that undivorced pearlite does not harden greatly in the tensile test as ferrite does. We found that the hardness of a broken tensile test piece, close to the fracture, was about twice as hard as the undeformed part in the case of "ingot iron," and 47 per cent. harder than the undeformed part in the case of steel of 0.11 per cent. of carbon,¹⁵ but the increase was very slight in the case of steel of 0.92 per cent. of carbon, when its pearlite was undivorced.

Increase of the Brinell Hardness Caused by Retarding the Brinell Test

Material	Brinell Hardness		Percentage Increase Caused by the Retardation
	Under the Usual Direct Test	Under a Test Greatly Retarded	
Hadfield's manganese steel quenched from 1,100°.	170	202	19
"American ingot iron," carbon 0.01 per cent.	73	99	37
Eutectoid steel, carbon 0.92 per cent., air cooled from 900°.	259	259	0

¹⁵ *Proceedings of the American Society for Testing Materials*, vol. xiv, pt. 2, p. 28 (1914).

Cost Factors in Coal Production

Discussion of the paper of WILLIAM H. GRADY, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 101, May, 1915, pp. 1035 to 1064.

HOWARD N. EAVENSON, Gary, W. Va. (communication to the Secretary*).—Several members have asked the tonnage loaded per day per miner upon which the author based his figures. For the year 1913 at the mines of the United States Coal & Coke Co., for an output of 3,530,390 net tons, 14.45 tons per working day per loader was averaged, and for the year 1914, for a total output of 1,835,692 net tons, the figure was 15.52 tons per working day per loader. In addition to this, the miner cleaned his coal and gobbled the refuse, an average of at least 15 per cent. of the coal loaded, timbered the place, laid the track, and drilled and loaded his holes.

The reply to the query of E. W. Parker about the average production per man will depend upon what men are included in the number used. For all men employed, including shop, power plant, superintendents, clerks, engineers, and construction forces, the average production per man per day was, in 1913, 4.56 net tons, and in 1914, 5.49 net tons. As many of the men employed and whose names are on the roll do not work every day, the production per man working per day will average about 20 per cent. higher than this, and if the men actually working in and around the mines only are included the average production will be at least 50 per cent. more than the figure given.

For the various depths of cover encountered in our mining to date the following widths of rooms and pillars are observed, and experience has shown that these are ample.

Depth of Cover, Feet	Width of Rooms, Feet	Width of Pillars, Feet
Up to 300.....	18 to 24	42 to 36
300 to 500.....	18 to 24	57 to 51
Over 500.....	18 to 24	72 to 66
Up to 300.....	36 to 40	44 to 40
300 to 500.....	36 to 40	54 to 50
Over 500.....	36 to 40	64 to 60

To Jan. 1, 1915, the percentages of the areas covered by our 12 mines, developed and worked out, were as follows:

COST FACTORS IN COAL PRODUCTION

Areas Developed, Per Cent.		Areas Worked Out, Per Cent.	
By headings.....	32.4	By headings.....	32.4
In rooms.....	16.4	Chain pillars.....	2.9
In room pillars.....	28.1	Rooms.....	36.7
In barrier pillars....	23.1	Room pillars.....	28.0
100.0		100.0	

For any depth of cover encountered so far in mining in the Pocahontas region, Mr. Storr's statement about the proper amount of seam to be removed in first mining is undoubtedly correct, and the sizes of pillars experience has shown to be the most economical are much larger than the sizes actually required to support the overhanging strata. The first principle of any economically planned and managed coal mine should be to drive rooms only as they are actually needed and to begin removing the pillars as soon as the rooms have reached their limits, and it is unfortunate that the limitations of output, labor supply, dirty coal, etc., too frequently prevent this being done, even when so planned.

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Next in line comes the description of the "goods" in the advertisement, and from that brief description the reader is in a position to judge whether or not it will meet his needs and whether it is desirable or necessary to inquire for detailed information.

It is assumed that every manufacturer makes an honest effort to describe his product in the limited advertising space at his disposal. At that, the reader knows full well that an inquiry will bring literature, etc., giving detailed information.

As a rule the manufacturer goes to considerable expense to gather data for the convenience of purchasers of his product, and the average reader realizes that it is hardly possible to fully describe the article advertised in the space the manufacturer's pocket-book can afford. Hence the inquiry for a full description.

In the engineering field particularly, the reader realizes the value of the advertising pages for keeping him in touch with the newest and best equipment for performing the tasks his profession requires. The more fully he recognizes the information to be obtained from advertisements, the more up-to-the-minute he becomes.

While the writer is confident that the readers of the *Bulletin* of the A. I. M. E. are "up-to-the-minute," he also calls attention to the fact that the advertisers in the *Bulletin* are representative firms, whose descriptive literature gives in detail information of value. Their advertisements invite your request for their catalogues, etc.

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Greatest drilling efficiency depends upon knowledge of the rock, selection of the machine drill (whose dead time in handling and mounting is low and capable of keeping pace with the drill bit), and the drill bit itself.

In the hand-hammer class there are two types of machines, the "hand rotated" or plugger drill, and the "self rotated" or Jackhamer drill. Hand-rotated drills are confined to comparatively shallow work, pop shooting, etc., and even here they are rarely used where other requirements make it necessary to employ the self-rotated drill for other work



These drills are extremely light and need no mounting, weight ranging from 20 to 50 lb. Air consumption and first cost are low, and are the determining factors with small operators.

The self-rotated or Jackhamer type is used extensively for drilling down-holes in shafts, trenches, for taking up bottom, tunnel trimming, etc. A record of its performance is given in this article. Weight of this drill is as low as 18 to 20 lb. Average weight, however, ranges from 40 to 90 lb. Points in its favor are: low air consumption, automatic rotation, hole-cleaning device provided, and rapid drilling.

Among its accomplishments may be noted the following: American Zinc Co.'s shaft, average of five Jackhamer drills sank a total of 284 ft. in 13 weeks. Size of shaft was 7 by 22 ft. Weekly average footage, 21.9.

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BULLETIN, A. I. M. E.—INDUSTRIAL SECTION

Walter F. Patterson, Jr., contractor, sank 253 ft. with three machines from March 2 to June 8, averaging weekly 18 ft., which includes time of timbering. This was in one of the hoisting shafts of the Houston Collieries Co. at Carswell, W. Va. Shaft size overall was 24 ft. 4 in. by 16 ft. 4 in.

In the shaft of the Butte Alex. Scott Copper Co. at Butte, Mont., 101 ft. was sunk in 89 shifts with two drills. Size of shaft was 7 by 14 ft. each of two compartments. Upkeep expense in this case was but \$2 per month per machine.



These performances are interesting when compared with work accomplished with tripod drills, formerly employed in this class of work.

The stoper drill may be said to have a field distinctively its own. Self-feeding up to the rock, and built in hand-rotated and self-rotating types. Experience with the latter type has been comparatively recent. Advantages are: absence of mounting, ease of operation, rapid drilling, and close-quarters operation.

Typical cases of operation are as follows: A large Lake Superior mine recently drove 400 lin. ft. of raise in a total of 50 shifts with two stopers

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operated by one man each. Work consisted of driving four raises 100 ft. each. Timbering unnecessary other than occasional sprags by means of which the men reached the working places.

In a test conducted on the Port Richmond job, California, a stoper averaged $70\frac{3}{4}$ in. of drill hole in $19\frac{1}{2}$ min.

The advantages of the hammer drill are: ease of handling, mounting, and chucking steel; rapid drilling for comparatively deep and large diameter holes; laying the dust by means of the water feature, and the admission of air and water to the bottom of drill hole for cleaning purposes; close-quarters use.



Among its many accomplishments may be cited the case of the Locust Mt. Coal Co.'s tunnel at Shenandoah, Pa., where Dolan Bros., contractors, drove 259 ft. in 14 days last October, working three 8-hr. shifts per day, and using two Leyner drills in sandstone. The last 14 days a progress of 254 ft. was made, working in hard conglomerate rock.

In our August number we shall describe the use of drill bits under special and difficult conditions.

Trade Note

The Nordberg Manufacturing Co., of Milwaukee, Wis., announces that E. W. Swartwout, formerly of the Chicago office, will be associated after June 1 with Mr. MacLaren, in the New York office of the company. Enlarged offices have recently been taken in the new Equitable Bldg. 120 Broadway, New York. The Chicago office will be in charge of John E. Lord.

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Derrick Attachment for Excavating

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Its range is from the base of the derrick to the end of the boom, and below the base of the derrick the length of the dipper arm.

If deep digging is necessary below the base of the derrick the dipper arm may be changed from a short to a long arm.

In operating, the digging line drum is gently slacked until the carriage rolls down the boom, bringing the shovel down to the desired position, and then released entirely, allowing the dipper to swing back under the boom; the large end of the cam operates the grip and at the same time the operator lowers the boom, thus bringing its entire weight on the

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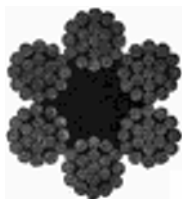
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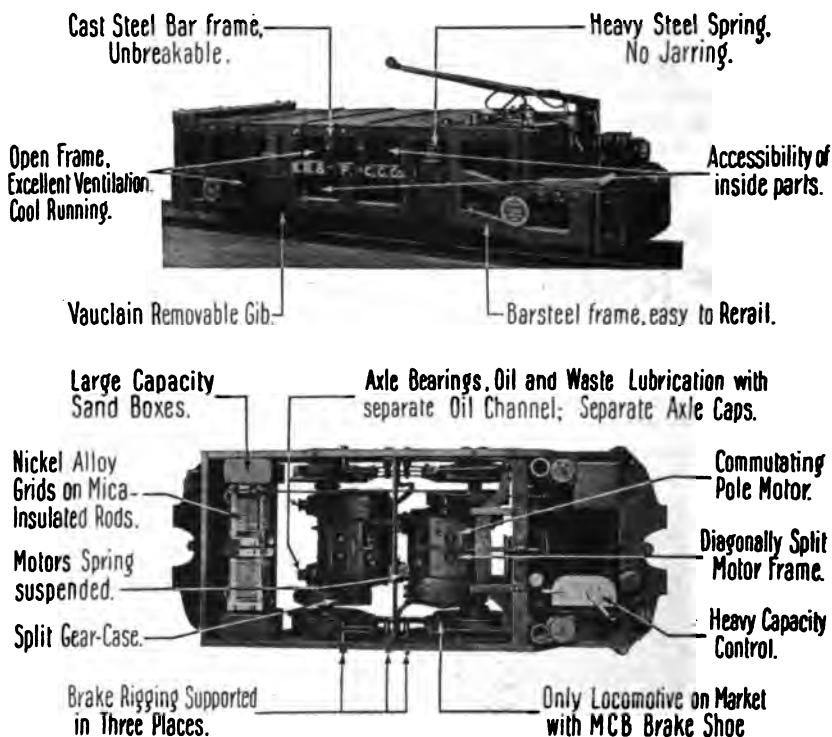
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AUGUST, 1915



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PUBLISHED MONTHLY

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Industrial Section	Interpolated in Advertising Section.

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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 104

AUGUST

1915

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Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

SAN FRANCISCO MEETING

Thursday, Sept. 16, to Saturday, Sept. 18, inclusive, 1915

Committee on Arrangements

CHARLES W. MERRILL, *Chairman*,
EDWARD H. BENJAMIN, H. C. HOOVER,
FRED. W. BRADLEY, W. C. RALSTON,
ABBOT A. HANKS.

Hotel Headquarters.—The headquarters of the Institute will be at the Bellevue Hotel, where most, if not all, of the technical sessions will be held. This hotel is conveniently located to transportation lines to the Exposition, all ferries, etc. The rates are from \$5 to \$10 per day, European plan, for two persons in one room with bath. Reservations should be made through the office of the Institute, or through Charles W. Merrill, Chairman, 121 Second Street, San Francisco. As the Local Committee has been obliged to make a guarantee of the use of a specified number of rooms to secure suitable reservations, it is hoped that members will patronize this hotel and make their reservations through the official channels.

Special Trains.—Arrangements for special trains from New York, and New Orleans, have been made by a Joint Committee on Transportation in New York City. The special trains will be run in conjunction with the American Society of Civil Engineers, the American Society of Mechanical

Engineers and the American Institute of Electrical Engineers. Already accommodations have been reserved for more than 150 persons, many of whom are members of this Institute. The special train from New York will leave at 7.45 p.m. on Sept. 9. A stop-over will be made at Niagara Falls, Colorado Springs, and at the Grand Canyon in Arizona. Arrangements have been made for the entertainment of the party at these points. The special train from New Orleans will leave that city on Sunday, Sept. 12, at 11.00 a.m. over the Southern Pacific lines, arriving in San Francisco on Wednesday, Sept. 15, at 1.00 p.m. Members may join either special train *en route*. Full details will be found in the circular already sent out, or may be obtained by application to the Secretary of the Institute.

Rates for these Trips.—The fare from New York to San Francisco, and return by any other route except *via* Portland, Ore. (good for three months), is \$98.80. The return trip *via* Portland is \$17.50 extra. The side trip from Williams to Grand Canyon is \$7.50. Pullman rates from New York to San Francisco, one way, lower berth \$22, upper berth \$17.60, compartment \$62 (one and one-half railroad fares required for use of one person), drawing room \$77 (two railroad fares required).

Reservations for this train are being made by G. S. Harner, Passenger Agent, New York Central Lines, 1216 Broadway, New York City, and application for reservations should be made directly to him. Mr. Harner will arrange for the transportation of those desiring to travel on this train, and for their return over any other line, or at any time specified.

The fare from New Orleans to San Francisco and return is as follows: Round-trip fare, New Orleans to San Francisco and return, good for three months, \$57.50, going and returning over the same route, or going *via* Southern Pacific and returning *via* Santa Fé; round-trip fare, New Orleans to San Francisco, returning to Chicago, \$62.50. Pullman rates, New Orleans to San Francisco, one way, lower berth \$11.50, upper berth \$9.20, drawing room \$41 (two railroad fares required), compartment \$32.50 (one and one-half railroad fares required for use by one person).

Reservations for this trip may be made by addressing J. H. R. Parsons, General Passenger Agent, Southern Pacific Company, New Orleans, La. In asking for reservations the statement should be made that the request is in connection with the engineering party. Reservations will be made in one car until that is filled, and so on; and should there be a sufficient number, the party can have a special train.

Technical Visits.—The Local Committee of the four National Societies and of the International Engineering Congress has arranged a series of excursions in California of more than ordinary interest, including a trip around San Francisco Bay by boat for members of this Institute and their guests only, and including a visit to the Selby Smelting Works; inspection of the high-pressure fire system in San Francisco; the hydraulic dredging on the Sacramento and San Joaquin Rivers; gold dredging at Oroville; the gold mines of Grass Valley; and the oil fields at Coalinga.

Ladies' Committee.—The Ladies' Committee consists of the following members, to whom others may be added with the purpose of having an ample number to insure that visiting ladies shall see the best of the Exposition, in not too large parties:

MRS. FRED W. BRADLEY,
MRS. G. HOWELL CLEVENGER,
MRS. WILLIAM HAGUE,
MRS. ABBOT A. HANKS,

MRS. H. C. HOOVER,
MRS. CHARLES JANIN,
MRS. EDMUND JUESSEN,
MRS. CHARLES W. MERRILL,

MRS. JAMES C. RAY,
MRS. MARK L. REQUA,
MRS. R. A. RICKARD,
MRS. W. H. SHOCKLEY.

The tentative plans of the Ladies' Committee include an evening reception and ball, an afternoon lawn party, and automobile trips through Golden Gate Park and Presidio, besides, of course, the banquet, at which the presence of ladies is especially urged.

Entertainment.—The chief source of entertainment in San Francisco will be the Panama-Pacific International Exposition. As a special compliment to the Institute, **Friday, Sept. 17**, has been designated by the Exposition management as the **American Institute of Mining Engineers' Day**. A banquet will be held, probably at the Exposition, on the evening of that day, and the program of the meeting has been arranged so as to give ample opportunity for those who desire to spend a good deal of time at the Exposition.

The Exposition.—The Exposition at California is alone worth to the members of the Institute all effort that may be necessary to make this trip. Architecture, sculpture, landscape gardening, mural paintings, grouping, decoration and color, all combine to give beauty, grace and dignity to the marvelous grouping of buildings situated close to the Golden Gate in one of nature's most attractive beauty spots. When seen in the day time the Exposition is delightful, but all join in giving unstinted praise to the magnificence of the spectacle during the special illuminations at night.

Contributed by the world's best designers, the Palaces are grouped around a series of connecting courts in such a manner as to attain a result unparalleled. The grounds cover 635 acres at an average width of a half mile, and two and a half miles along the bay shore.

The State buildings of California, Pennsylvania, New York, Massachusetts, Virginia, Ohio, Illinois, Missouri and others, are imposing and characteristic. The exhibits, at least as far as Mining and Metallurgy are concerned, are the best that have ever been gathered together at any one place. Twenty-eight foreign countries are represented at the Exposition, and among these, the exhibits of Japan, Canada, and Argentine Republic are worthy of special mention. For its educational value, attractiveness, and as an example of what can be accomplished in the way of securing the co-operation of many diverse interests, the Palace of Mining and Metallurgy is worthy of the careful attention of every member of the Institute.

PROGRAM OF MEETING

Thursday, Sept. 16, 1915

Registration at Hotel Bellevue, on this and all days of the meeting.

10 a. m. Session. Charles W. Merrill presiding:

Address of welcome on behalf of California.

Reply by President Saunders of the A. I. M. E.

Underground Mining Systems of Ray Consolidated Copper Co., by LESTER A. BLACKNER

Some Problems in Copper Leaching, by L. D. RICKETTS

Notes on Homestake Metallurgy, by ALLAN J. CLARK

The Metallurgy of Gold in the Witwatersrand District, South Africa, by F. L. BOSQUI

2 p.m. Simultaneous Sessions on Gold and Silver, on Mining Geology and Mineralogy, and on Iron and Steel.

Session on Gold and Silver, F. Lynwood Garrison presiding:

Mill and Cyanide Plant of Chiksan Mines, Korea, by CHARLES W. DEWITT
Electric Furnace for Gold Refining at the Alaska-Treadwell Cyanide Plant,
by W. P. LASS

Cyaniding Practice of Churchill Milling Co., Wonder, Nev., by E. E.
CARPENTER

Zinc-Dust Precipitation Tests, by NATHANIEL HERZ

Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt,
by E. B. THORNHILL

Slime Agitation and Solution Replacement Methods, West End Mill,
Tonopah, Nev., by JAY A. CARPENTER

Amalgamation Tests, by W. J. SHARWOOD

A Rule Governing Cupellation Losses, by W. J. SHARWOOD

The Tonopah Plant of the Belmont Milling Co., by A. H. JONES

Session on Geology and Mineralogy:

The Geology of the Iron-Ore Deposits in and Near Daquiri, Cuba, by
JAMES F. KEMP

The Formation and Distribution of Residual Iron Ores, by C. L. DAKE

The Occurrence of Covellite at Butte, Mont., by A. PERRY THOMPSON

The Formation and Distribution of Bog Iron Ores, by C. L. DAKE

Geology of the Burro Mountains Copper District, New Mexico, by R. E.
SOMERS

Additional Data on Origin of Lateritic Iron Ores of Eastern Cuba, By
C. K. LEITH and W. J. MEAD

Method of Making Mineralogical Analysis of Sand, by C. W. TOMLINSON

The Formation of the Oxidized Ores of Zinc from the Sulphide, by Y. T. WANG

The Copper Deposits of San Cristobal, Santo Domingo, by THOMAS F.
DONNELLY

Session on Iron and Steel, Joseph W. Richards presiding:

Iron Ores of California and Possibilities of Smelting, by C. COLCOCK JONES
Conversion Scale for Centigrade and Fahrenheit Temperatures, by HUGH
P. TIEMANN

Manufacture and Tests of Silica Coke-Oven Brick, by KENNETH SEAVER

The Duplex Process of Steel Manufacture at the Maryland Steel Works, by
F. F. LINES

The Electric Furnace in the Foundry, by WILLIAM G. KRANZ

Commercial Production of Sound Homogeneous Steel Ingots and Blooms, by
E. GATHMANN

Suggestions Regarding the Determination of the Properties of Steel, by
A. N. MITINSKY

Friday, Sept. 17, 1915

9 a. m. Simultaneous Sessions on Electrometallurgy and on Petroleum and Gas:

Joint Session with the American Electrochemical Society, Lawrence Addicks, President of the A. E. S., presiding:

Metallurgical Industries as Possible Consumers of Electric Power, by
DORSEY A. LYON and ROBERT M. KEENEY

Melting of Ferro-Alloys in the Electric Furnace, by R. S. WILE

The Thermal Insulation of High-Temperature Equipment, by PERCY A.
BOECK

Radiography of Metals, by WHEELER P. DAVEY

Roasting and Leaching Concentrator Slimes Tailings, by LAWRENCE ADDICKS

Deposition of Copper from Leaching Solutions, by LAWRENCE ADDICKS

Hydro-electrolytic Treatment of Copper Ores, by ROBERT RHEA GOODRICH

Solution Stratification as an Aid to the Purification of Electrolytes, by
FRANCIS R. PYNE

Session on Petroleum and Gas, Arthur F. L. Bell presiding:

- The Occurrences of Petroleum in the Oil Fields of Eastern Mexico as Contrasted with those of Texas and Louisiana, by E. T. DUMBLE
 The Mexican Oil Fields, by L. G. HUNTLEY
 Gasoline from "Synthetic" Crude Oil, by WALTER O. SNELLING
 Correlation and Geological Structure of the Alberta Oil Fields, by D. B. DOWLING
 Oil, Gas, and Water Content of Dakota Sand in Canada and United States, by L. G. HUNTLEY
 Sliding Royalties for Oil and Gas Wells, by ROSWELL H. JOHNSON
 The Possible Occurrence of Oil and Gas Fields in Washington, by CHARLES E. WEAVER
 Petroleum as Fuel under Boilers and in Furnaces for Heating, Melting, and Heat Treatment of Metals, by W. N. BEST
 The Furbero Oil Field, Mexico, by E. DE GOLYER

2 p. m. Session on Mining, Milling, and Non-Ferrous Metallurgy, Karl Eilers presiding:

- Tramming and Hoisting at the Copper Queen Mine, by GERALD SHERMAN
 Ventilation of the Copper Queen Mine, by CHARLES A. MITKE
 Mine Pumping, by CHARLES LEGRAND
 Fire-Fighting Methods at Mountain View Mine, Butte, Mont., by C. L. BERRIEN
 Mining Conditions on the Witwatersrand, by W. L. HONNOLD
 Churn-Drilling Costs, Sacramento Hill, by ARTHUR NOTMAN
 The Application of the Apex Law at Wardner, Idaho, by FRED T. GREENE
 The Stresses in the Mine Roof, by R. DAWSON HALL
 Standardizing Rock-Crushing Tests, by M. K. RODGERS
 Rotary Desulphurizing Kilns, by S. E. DOAK
 Kick vs. Rittinger, by ARTHUR O. GATES
 The Concentrator of the Timber Butte Milling Co., Butte, Mont., by THEODORE SIMONS
 Hardinge Mill Data, by ARTHUR F. TAGGART
 The British Columbia Copper Co.'s Smelter, Greenwood, B. C., by FREDERIC K. BRUNTON
 The Salida Smelter, by F. D. WEEKS
 Lead Smelting at El Paso, by H. F. EASTER
 The Advantages of High-Lime Slags in the Smelting of Lead Ores, by S. E. BRETHEERTON
 The Mellen Rod-Casting Machine, by R. C. PATTERSON, JR.

Saturday, Sept. 18, 1915

- 10 a. m. Special Boat Trip around San Francisco Bay, and visit to Selby Smelting Works. (Members and guests of A. I. M. E. only.)
 10 a. m. The San Francisco High Pressure Fire System, and the Gas and Electric Works.
 10 a. m. Spring Valley Water Works.
 Start Friday night, 11 p. m. Gold Dredging at Oroville, and Hydro-electric Development at Las Plumas.
 Start Friday night, 12.45 a. m. The Grass Valley Mines and Hydro-electric Development.
 Start Friday night, 10.30 p. m. The Coalinga Oil Fields.

INTERNATIONAL ENGINEERING CONGRESS

Monday, Sept. 20, to Saturday, Sept. 25, 1915.

NOMINATIONS FOR OFFICERS

The co-operation of the members of the Institute is earnestly sought by the Committee on Nominations, recently appointed by the Board of Directors, in its work of formulating a ticket for officers and places falling vacant in the Board of Directors in February, 1916. The Committee must present the official ticket of nominations to the Board of Directors at its October meeting. Members are, therefore, urged to send promptly their nominations for officers and Directors in order that the Committee may have an opportunity to give them full consideration.

The officers to be elected at the annual meeting in February, 1916, are: One officer, known as Director and President. Two officers, known as Director and Vice-President. Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N. B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, William L. Saunders (not eligible for re-election).

Past President, Charles F. Rand.

Vice-Presidents, Thomas H. Leggett, District 0; Fred W. Denton, District 4.

Directors, John W. Finch, District 7; John H. Janeway, District 0; Edward P. Mathewson, District 5; Joseph W. Richards, District 2; George D. Barron, District 0.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, Crocker Bldg., San Francisco, Cal.

COMMITTEE ON NOMINATIONS: F. W. BRADLEY, *Chairman*.

LOUIS S. CATES,
R. C. GEMMELL,

JAMES F. KEMP,
FRANK A. ROSS,

FRANK M. SMITH,
ARTHUR THACHER.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period June 10 to July 10, 1915:

Milton R. Evans, Plymouth, Pa.
R. A. F. Penrose, Jr., Philadelphia, Pa.
Charles S. Burt, Ann Arbor, Mich.
Durward Copeland, Rolla, Mo.
J. R. Finlay, New York, N. Y.
Fred T. Williams, Salt Lake City, Utah.
D. W. Brunton, Denver, Colo.
Edwin E. Chase, Denver, Colo.

Francis Church, Lincoln, Reno, Nev.
John G. Worth, Albuquerque, N. M.
S. A. Taylor, Pittsburgh, Pa.
S. LeFevre, Forest Glen, N. Y.
Oliver Bowles, Washington, D. C.
Takeshi Kawamura, Tokyo, Japan.
Francis C. Phillips, Pittsburgh, Pa.
Nathaniel Herz, Lead, S. D.

Max Roesler, Instructor in Geology at Yale, is the recipient of the Emmons Memorial Fellowship.

J. H. Batchellor has accepted a position as Superintendent of a mine and mill of the U. S. S., R. & M. Exploration Co., at Joplin, Mo.

W. C. Coffin has been elected President of the University of Pittsburgh General Alumni Association for the ensuing collegiate year.

H. D. Williamson, who for some years was in Japan, has left that country and is now located in England, where he will practice as a consulting and inspecting engineer. His address is 83 Hayes Road, Bromley, Kent, England.

Charles H. Schmalz, for the past six years Master Mechanic for the Anaconda Copper Mining Co., Boston & Montana Reduction Department, Great Falls, has been appointed manager of the Billings Foundry & Mfg. Co. of Billings, Mont.

Irving A. Palmer has accepted the position of Superintendent of the United States Smelting Co.'s plant at Altoona, Kan.

T. N. Havlin has resigned his position as Professor of Chemistry and Metallurgy at the Kansas State School of Mines and Metallurgy, to take up private work at Weir, Kan.

Ralph S. Cochran has accepted the position of Metallurgist of the Republic Iron & Steel Co., Youngstown, Ohio.

Stuart B. Marshall was awarded a post-graduate degree of Master of Science by the Academic Board and the Board of Visitors of the Virginia Military Institute.

S. H. Sherman has been made Superintendent of the Christmas mine of the American Smelting & Refining Co., Christmas, Ariz.

Arthur P. Allen is engineer with the Winona Copper Co., Winona, Mich.

Charles N. Gould, formerly Professor of Geology at the State University of Oklahoma, and first Director of the Oklahoma Geological Survey, from which position he resigned in 1911 to enter consulting practice, now has an office at 1218 Colcord Bldg., Oklahoma City, Okla. He is specializing on the geology of petroleum and natural gas, particularly in the Mid-Continent field and the Gulf Coast of Texas and Mexico.

John A. Fulton, Superintendent at Melones mine, Melones, Cal., was presented with a daughter on June 12.

Ellison C. Means has resigned as President and General Manager of the Low Moor Iron Co., Low Moor, Va. His address for the present is, Box 99, Rockville, Md.

A. Faison Dixon is now Field Manager for the Colon Development Co., Ltd., with headquarters at Encontrados, Estata Zulia, Venezuela.

JOSEPH A. HOLMES

Dr. Joseph A. Holmes, Director of the U. S. Bureau of Mines, died at his home in Denver, Colo., on the night of July 12, 1915, thus closing a life of unusually courageous, energetic, and fruitful activity in geology and mining. A biographical notice of Dr. Holmes will appear in a subsequent number of the *Bulletin*.

ENGINEERS' CLUB OF BALTIMORE

The Annual meeting of the Engineers' Club of Baltimore resulted in the election of Oscar F. Lackey, President; W. Watters Pagon, Vice-President; John N. Mackall, Chairman of House Committee; R. Keith Compton, William Walter Crosby, and Ezra B. Whitman, members of the Board of Directors.

The Board elected William D. Janney to succeed himself as Secretary-Treasurer, and the President appointed chairmen of the standing committees as follows: Finance, William F. Strouse; Publication, Herman Eisert; Publicity, Layton F. Smith; Library, C. J. Tilden; Membership, Edward F. Ruggles; Tellers, J. Talbott Todd; Auditors, Albert H. Wehr.

The first meeting of the Club for the ensuing season will be held on the first Thursday in September. Communications should be addressed to William D. Janney, Secretary-Treasurer, 4-6 West Eager St., Baltimore, Md.

POSITIONS AVAILABLE

During the past few weeks the Institute has been in receipt of a number of inquiries for iron and steel metallurgists. The Institute's office has not been in a position to fill all of these positions with exactly the type of man required. Iron and steel metallurgists available for vacancies are, therefore, requested to communicate with the Employment Department.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Associate Professorship in Department of Metallurgy of a Northwestern university for technically trained man with considerable practical experience, to devote half his time to research and half to teaching. Salary \$2,000 for eleven months' service. No. 40.

Associate Professorship in charge of Department of Mining for technically trained man with considerable practical experience, to devote half his time to research and half to teaching. Salary for eleven months' service \$2,000. No. 41.

Associate Professorship of Mining for man with some experience, about 30 years of age; to have charge of summer school which will occupy two periods of two to three weeks each, but not consecutively. Salary \$1,800 per annum. No. 42.

A steel-casting plant in the middle West desires to secure an experienced man as blower in a side-blow converter plant. No. 43.

Service in Asiatic countries and Turkey as representative of large oil interests. Duties range from office work and sales management to organization and development of agencies. Applicants must be unmarried and between 21 and 28 years of age. Foreign language is desirable, but not essential. Applications should be in writing. No. 44.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Position wanted by member, technical graduate, B. S. and E. M., seven years' experience, six in anthracite coal fields, and one in teaching in middle West; 29 years old, married. Available immediately; will go anywhere. Reason for change, discontinuance of work at present engaged in. References. No. 236.

Member, technical graduate, aged 36. Experienced copper metallurgist and mining geologist. Familiar also, through 15 years' broad practical experience, with mining, cyanidation, concentration (including oil flotation), metallurgical accounting, and mine and plant management. Best references. No. 237.

Member, technical graduate, 11 years' experience in gold, silver, and copper mining and milling. At present in charge of large property in Mexico but wishes change owing to the present Mexican situation. Will go anywhere as manager or superintendent or examination work. Minimum salary \$4,000. No. 238.

Member, technical graduate; having two weeks' vacation in September, desires to communicate with some engineer who has examination work during that period, and needs an assistant. No. 239.

Member with 20 years' experience in the oil fields of the southern part of the Great Plains, is available for the examination of oil and gas properties. No. 240.

Member desires position as inspector or superintendent of coal mines; 23 years' practical experience; experienced in gaseous mines, also served as State Inspector; 41 years of age, strictly temperate, married. Satisfaction guaranteed. No. 241.

Member, young man, desires foreign opening. Prefers Korea, Siberia, Manchuria, but would go anywhere. American-born, Scotch-American descent; technical education, seven years' experience assaying, sampling, exploration work, cyaniding. Good mineralogist. Quick on languages. Excellent references. No. 242.

Member, technical graduate, at present employed as assistant superintendent by one of the world's largest copper companies, desires change. Has had exceptional experience with basic-lined converters, blast furnaces, reverberatories, and roasters; has also had experience in leaching and mill work. Can furnish best of references. No. 209.

SOUTHERN CALIFORNIA LOCAL SECTION*Executive Committee*SEELEY W. MUDD, *Chairman*C. COLCOCK JONES, *Vice-Chairman*FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg.,
Los Angeles, Cal.

RALPH ARNOLD

A. B. CARPENTER

The Second Annual Meeting of the Southern California Local Section of the American Institute of Mining Engineers was held at 300 Germain Bldg. on June 18, 1915, at 8 P.M.

Present were: Messrs. C. Colcock Jones, Carpenter, Merrill, Baumgarten, Staunton, Hodges, Holland, and Durell.

In the absence of the Chairman, Mr. Jones presided. The tellers appointed to canvass the vote for Section officers reported 45 ballots cast. The following gentlemen were announced as duly elected to the several offices:

Chairman, SEELEY W. MUDD,
Vice-Chairman, C. COLCOCK JONES,
Secretary-Treasurer, F. J. H. MERRILL,
Members of Executive Committee,

RALPH ARNOLD

A. B. CARPENTER

A proposed Amendment to the By-Laws regarding future elections was discussed, and carried by two-thirds vote. The Secretary-Treasurer then reported on the work of the Section and its financial condition.

It was moved, as an Amendment to the By-Laws, to be acted on at the next meeting, that the Executive Committee be increased to seven members, and that the present Committee be empowered to fill the two additional places.

After informal discussion of Section business, the meeting adjourned.

F. J. H. MERRILL, *Secretary*.

UTAH LOCAL SECTION*Executive Committee*ROBERT C. GEMMELL, *Chairman*C. W. WHITLEY, *Vice-Chairman*ERNEST GAYFORD, *Secretary*, 159 Pierpont Ave., Salt Lake City

WALTER FITCH

WILLIAM WRAITH

Under date of June 25, the Utah Local Section of the Institute announced an excursion to the Tintic mining district, to take place on July 15, returning on July 16. The program comprised a tour of inspection of the entire district by automobile and Knight Railroad and a session for the reading of technical papers, followed by a social session. The papers to be presented were:

Geology of Ore Deposits and Geological Methods Applied to Mining in the Tintic District, Utah, by G. W. CRANE.

The Zinc Problem in the Treatment of Low-Grade Zinc and Zinc-Lead Ores, by D. A. LYON.

ERNEST GAYFORD, *Secretary*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
 AMERICAN SOCIETY OF MECHANICAL ENGINEERS
 AMERICAN INSTITUTE OF MINING ENGINEERS
 UNITED ENGINEERING SOCIETY
 WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining, Metallurgy, and Chemistry

MINING INDUSTRY IN THAT PART OF NORTHERN ONTARIO SERVED BY THE TEMISKAMING AND NORTHERN ONTARIO RAILWAY, 1914. Toronto, 1915. (Gift of Temiskaming and Northern Ontario Railway Commission.)

HISTORIA DEL TRATAMIENTO METALÚRGICO DEL AZOGUE EN ESPAÑA. By D. Luis de la Escosura y Morrogh. Madrid, 1878. (Gift of O. H. Hahn.)

JAARVERSLAG VAN DE WINNING, HET VERVOER EN DEN VERKOOP VAN BANKA-TIN, 1913. Batavia, 1915.

ENZYKLOPÄDIE DER TECHNISCHEN CHEMIE. Vol. 2. By Fritz Ullmann. Berlin, 1915.

Geology and Mineral Resources

CORUNDUM, ITS OCCURRENCE, DISTRIBUTION, EXPLOITATION AND USES. (Memoir 57, Canada Department of Mines.) Ottawa, 1915.

PETROLEUM AND NATURAL GAS RESOURCES OF CANADA. Vol. 1. Ottawa, 1914.

COAL FIELDS AND COAL RESOURCES OF CANADA. (Memoir 59, Canada Department of Mines.) Ottawa 1915.

RED IRON ORES OF EAST TENNESSEE. (Bull. 16, Tennessee Geological Survey.) Nashville, 1913.

GEOLOGY OF THE COUNTRY BETWEEN KALGOORLIE AND COOLGARDIE. (Bull. 56, Western Australia Geological Survey.) Perth, 1914.

ANLEITUNG ZUM GEOLOGISCHEN BEOBSACHTEN, KARTIEREN UND PROFILIEREN. Von Hans Höfer von Heimhalt. Braunschweig, Vieweg & Sohn, 1915. (Gift of Author.)

[NOTE.—A practical work describing the work of the field geologist and cartographer.—W. P. C.]

GEOLOGY OF NEW JERSEY. (Bull. 14, New Jersey Geological Survey.) Union Hill, 1915.

GEOLOGICAL RECONNAISSANCE OF A PORTION OF THE MURCHISON GOLDFIELD. (Bull. 57, Western Australia Geological Survey.) Perth, 1914.

GEOLOGY AND MINING AT SANDSTONE AND HANCOCK'S, EAST MURCHISON GOLDFIELD NOTES. (Bull. 62, Western Australia Geological Survey.) Perth, 1914.

General

BIBLIOGRAPHIE DER DEUTSCHEN ZEITSCHRIFTEN LITERATUR. Band XXXV, 1914. Leipzig, 1915.

BY-PRODUCT COKING. References to books and magazine articles. Carnegie Library of Pittsburgh. Pittsburgh, 1915. (Gift of Carnegie Library of Pittsburgh.)

CARNEGIE STEEL CO. Pocket Companion for Engineers, Architects, and Builders, containing useful information and tables appertaining to the use of steel. Ed. 17. Pittsburgh, 1915.

CARNEGIE STEEL CO. Standard Specifications. Ed. 5. Pittsburgh, 1915. (Gift of Carnegie Steel Co.)

DIRECT ACTING STEAM PUMPS. By Frank F. Nickel. New York, McGraw-Hill Book Co., 1915. (Gift of Author.)

[NOTE.—The author, while realizing the encroachment of the centrifugal pump in the field, has made it his object to preserve the history of the direct-acting pump. He therefore devotes his first chapter to the development of this mechanism. He discusses performance factors, and describes typical pumps, suited to various service conditions.—W. P. C.]

GENERAL FACTORY ACCOUNTING. By F. H. Timken. Chicago, 1914.

Company Reports

BUTTE & SUPERIOR COPPER CO., LTD. Annual Report, 1st-3d. New York, 1912-14.

BUTTE & SUPERIOR COPPER CO., LTD. Fifth Quarterly Report, Mar. 31, 1915. (Gift of Butte & Superior Copper Co.)

Trade Catalogues

BONNEY-FLOYD Co., Columbus, Ohio.

Bonoyd manganese steel. Car couplings, clevises, pins.

Car mover.

Car replacer.

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill.

Bull 34-U. Instructions for installing and operating "Chicago Pneumatic" Class N/SO fuel oil driven compressors. May, 1915.

Bull. 34-X. Class A-G "Giant" gas and gasoline engines, May, 1915.

THE CUTLER-HAMMER MFG. CO., Milwaukee, Wis. Booklet D. The Thomas Meter. June, 1915.

DUPONT DE NEMOURS POWDER CO., E. I., Wilmington, Del. Dupont Magazine. June and July, 1915.

HADFIELDS, LTD., Sheffield, England. No. 129. Modern high-class crushing machinery.

HUMBOLDT MANUFACTURING CO., Chicago, Ill. Humboldt Recorders. General description.

INGERSOLL-RAND CO., New York, N. Y. Form '3031. "Ingersoll Rogler class FR-1" air compressors. May, 1915.

JEFFERY CO., THOMAS B., Kenosha, Wis. Jeffery Quad. Truck Catalog F-3c. 11 pp.

NATIONAL TRANSIT CO., Oil City, Pa.

Bull. 104. Steam pumps. General service.

Bull. 105. Steam pumps. Slush and valve-pot type.

Bull. 401. Vertical gas engine, air cooled, four cycle, $3\frac{1}{2}$ B. H. P.

PICHER LEAD CO., Chicago, Joplin, New York. Protective coatings for structural metals. 46 pp.

SPARTA IRON WORKS CO., Sparta, Wis. Sludge bucket. July, 1915.

STROUD, E. H. & Co., Chicago, Ill.

Bull. 101. Air separation pulverizer.

Bull. 103A. Powdered coal.

Bull. 102A. Screen separation mill

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period June 10 to July 10, 1915:

Members

- AGTHE, FRED. THOMAS, Quarry Supt., Atlas Portland Cement Co., Hannibal, Mo.
 ARTHUR, WILLIAM SEYMOUR, Project Mgr., U. S. Reclamation Service, Williston, S. D.
 ASH, SIMON HARRY, Deputy State Mine Inspector, 503 Alaska Bldg., Seattle, Wash.
 BALDWIN, C. KEMBLE, Second Vice-Pres. and Chicago Mgr.,
 Robins Conveying Belt Co., 1315 Old Colony Bldg., Chicago, Ill.
 BOGGS, WILLIAM BRENTON, Met. Douglaston, L. I., N. Y.
 BROWN, RALPH DARE, Min. Engr., Asst. Genl. Supt., O'Gara Coal Co., Harrisburg, Ill.
 COLE, A. NEWTON Genl. Supt., Pennsy Coal Co., Clarion, Pa.
 DAVIS, JOSEPH D., Chem. Bureau of Mines, Washington, D. C.
 DEAN, DUDLEY STUART, Treas., Keweenaw Land Association, Ltd.,
 Newport Land Co., 87 Milk St., Boston, Mass.
 DOLAN, HUGH, Contractor (Mining) 1755 W. Market St., Pottsville, Pa.
 DOWNING, FRANK E., Asst. Mgr., Iron Mines, Shenango Furnace Co., Chisholm, Minn.
 FILTEAU, CHARLES ALEXANDER, Min. Engr. 157 Lexington Ave., New York, N. Y.
 FITCH, HOWARD W., Min. Engr., Asst. to Geol., Chief Consolidated
 Mining Co., Eureka, Utah.
 GEHRES, GEORGE W. Genl. Supt., Republic Iron & Steel Co., Republic, Pa.
 GODSHALK, ELMER GRANT Supt., Bartlesville Zinc Co., Collinsville, Okla.
 HALL, CHARLES JAMES, Met.; Smelter Supt., Spassky Copper Mine, Ltd.,
 Karagandy, Akmolinsk Province, Siberia, via San Francisco or Puget Sound.
 HAZELTINE, BENJAMIN PRESCOTT, Rolling Mill Supt., National Tube Co.,
 Wheeling, W. Va.
 HIGGINS, ARTHUR HOWARD, Met. East Rockaway, L. I., N. Y.
 HOLMAN, STEPHEN ADAIR, JR., Mine Supt., Balaklala Cons. Copper Co.,
 Coram, Shasta Co., Cal.
 HOLSTEIN, LEON STUART, Chem. New Jersey Zinc Co., Palmerton, Pa.
 JAHN, GUSTAVE A. 141 E. 19th St., Brooklyn, N. Y.
 KIRCHEN, JOHN G., Mine Mgr., Tonopah Extension Mining Co., Tonopah, Nev.
 KLEPETKO, ERNEST, Min. and Met. Engr. 706 Oak St., Anaconda, Mont.
 MEREDITH, WILLIAM F., Pres., Titanium Alloy Mfg. Co., 15 Wall St., New York, N. Y.
 MOSONYI, EMIL, Technical Director, Salvador Deep Well Boring Co.,
 San Salvador, Salvador.
 NIEMAN, HENRY WILLIAM, Min. Engr., Arctic Placer Mining & Milling Co.,
 Nome, Alaska.
 SCHROEDER, ALBERT HEINRICH Instructed to hold all mail.
 SMITH, HOWARD C. Millman, Portland Gold Min. Co., Cripple Creek, Colo.
 STURTEVANT, THOMAS E., Chief Engr., McKiernan-Terry Drill Co.,
 233 Broadway, New York, N. Y.
 TERRILL, ARTHUR CLARK, Min. Engr., Geol. 1123 Chestnut St., Glendale, Cal.
 TICKNER, FRANK W. Genl. Supt., Valley Mould & Iron Co., Sharpsville, Pa.
 VAN HORN, EDWARD A. Supt. Susquehanna Coal Co., Shaft, Schuylkill Co., Pa.
 WAITE, ALLAN GRIGGS, Mill Metallurgy, Flotation Box 628, McGill, Nev.
 WARREN, HARRY MUNSEN, Elec. Engr., Delaware, Lackawanna & Western R. R. Co.,
 Scranton, Pa.
 WEINIG, ARTHUR JOHN, Cyanide Mill Supt., Liberty Bell Gold Min. Co.,
 Telluride, Colo.
 WHITAKER, CHARLES NEWTON, JR., Min. Engr., Apartado 5, Santa Barbara,
 Chih., Mexico.
 WHITMAN, ALFRED RUSSELL, Min. Geol. Timmins, Ont., Canada.
 WILSON, FRANK LAKE, Engr., Atkins, Kroll & Co., 311 California St.,
 San Francisco, Cal.
 WOLFE, CONRAD, Pres. and Genl. Mgr., United Copper Mining Co., Chewelah, Wash.

Associate Member

- ELLIOTT, WILLIAM RUPERT, Treas., J. George Lyner Engineering Works, Denver, Colo.

Junior Members

BLAYLOCK, DANIEL WEBSTER, Min. Engr.	Instructed to hold all mail.
BRADT, MAURICE L., Student	935 S. Fourth St., Saginaw, Mich.
CUTTING, GEORGE WALFORD	Florence, Colo.
DARRINGTON, PAUL NEWMAN	2815 St. Paul St., Baltimore, Md.
GRANTING, HUGO GEORGE, Student	Box 373, Florence, Colo.
KERR, DUNCAN MACMILLAN, Engr. Dept., Anaconda Copper Mining Co.,	Anaconda, Mont.
LOERPABEL, W. HARRISON, Student	Care Joseph Gillette, Mazomanie, Wis.
MCCUTCHEON, KENNETH CHARLES, Student, 112 Harvard Road, Thornburg,	Pittsburgh, Pa.
MAXON, WALTER LANDON	60 Greenbush St., Cortland, N. Y.
NEELD, HARPER C., Dept. of Met. Research, University of Utah, Salt Lake City, Utah.	
RAIBER, NICOLAUS H.	528 Dawson Ave., Bellevue, Pa.
STIFEL, CARL GODFRIED, Student	2007 Hebert St., St. Louis, Mo.

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period June 10 to July 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

P. R. Allen, McAlester, Okla.

Proposed by Eugene McAuliffe, Cyrus Garnsey, Jr., Carl Scholz.

Born 1878, New Orleans, La.

Present position: Pres., McAlester Coal & Coke Co.

Jesse Oatman Betterton, Collinsville, Ill.

Proposed by William E. Newman, James A. Caselton, Herman Garlich.

Born 1884, Kouts, Ind. 1905, Columbus, Neb., High School. 1909, B. S. in Met. Engrg.; 1912, Met. Engr., So. Dakota State School of Mines. 1905-09 (vacations), Practical mining at Butte, Mont., and Lead, S. D. 1909-10, Assaying and surveying, Leesburg Mining Co., Leesburg, Ida.

Present position: Supt. Basic-Sulphate and Blast-Furnace departments, St. Louis Smelting & Refining Co.

David Montgomery Claghorn, Roslyn, Wash.

Proposed by Raymond P. Tarr, Edward V. d'Inwilliers, Clarence R. Claghorn.

Born, 1891, Birmingham, Ala. 1903-05, Hill School, Pottstown, Pa. 1906-10, Taft School, Watertown, Conn. 1910-11, Sheffield Scientific School, New Haven. 1911-13, Columbia School of Mines, New York. 1913-14, Practical underground mining work, Northwestern Improvement Co., Red Lodge, Mont.

Present position: 1914 to date, Coal-Washery Foreman, Roslyn Fuel Co.

Claude H. Cooper, Hubbell, Mich.

Proposed by D. H. Ladd, William H. Bassett, J. F. B. Erdlets, Jr.

Born 1880, Hancock, Mich. 1899-1903, Mass. Inst. of Technology. 1903-04, Quincy Smelting Works, Hancock, Mich.

Present position: 1904 to date, Asst. Supt., Calumet & Hecla Smelting Works.

Francis A. Crampton, Goodsprings, Nev.

Proposed by W. G. Haldane, Frank A. Keith, Seeley W. Mudd.

Born 1888, New York City. 1899, Columbia Institute, New York City, grade school. 1906, Palmer Institute, Lakemont, N. Y., high school. 1907-08, Surveying, Boston Consolidated Mining Co. 1908-11, Mgr., Arizona Copper Belt Min. Co. 1911, White-lead salesman, Matheson Lead Co., lead corrodors. 1912, Millman, American Rare Metals Co. 1912, Solution man, Primos Chemical Co. 1912-13, Ceramics Department (under Prof. G. M. Butler) of Colorado Geological Survey. 1914, Mill Shift Boss, Hilltop Mining Co.

Present position: Asst. Supt., Platinum Gold Mining Co., Boss Mine.

Fred W. Crosley, Cripple Creek, Colo.

Proposed by F. K. Brunton, J. W. Finck, Irving T. Snyder.

Born 1868, Centre Point, Ind. 1892-93, School of Mines, Colorado (did not graduate). Practical experience and association with many other engineers. 1894, Lloyd and Rigney. 1895, Lloyd, Rigney, and George W. Lloyd. 1896, T. P. Rigney. 1896-1915, U. S. Mineral Surveyor and mining engineer. 1906-13, Mary McKinney G. M. Co.

Present position: 1907 to date, Min. Engr., Vindicator Cons. G. M. Co.; Cresson Consolidated G. M. Co.; Gold Dollar Consolidated Min. Co.; Henry Adney G. M. Co.; United Gold Mines Co.

Isador Aaron Ettlinger, Superior, Ariz.

Proposed by Walter H. Aldridge, Fred B. Ely, W. C. Browning.

Born, 1886, Joliet, Ill. 1904, Grad., Joliet High School. 1905, Chicago Univ. 1909, Michigan College of Mines; M. E. and B. S. 1909, Min. Engr., New Keystone Copper Co., Miami, Ariz. 1910, Min. Engr., Spurr & Cox, El Paso, Tex. 1911, Assayer, Socorro Min. Co., Mogollon, N. M. 1912, Assayer, Inspiration Copper Co., Miami, Ariz.

Present position: 1912 to date, Min. Engr., Magma Copper Co.

Ben Benight Hood, Chrome, N. J.

Proposed by Ralph W. Deacon, Francis R. Pyne, Robert M. Draper.

Born 1886, Manhattan, Kan. 1893-95, Public schools. 1905-08, Michigan College of Mines; M. E. 1909-13, Chief Engr., Winona Copper Co. 1913, Draftsman, British America Nickel Corp., Sudbury, Ont. 1914, Mill Supt., Moose Mountain, Ltd., Sellwood, Ont.

Present position: Boiler Room Supt., United States Metal Refining Co.

Takeo Igawa, Ray, Ariz.

Proposed by Louis S. Cates, W. S. Boyd, Lester A. Blackner.

Born 1890, Japan. 1902-06, Iwakuni Military Academy, Japan. 1907-09, Hyde Park High School, Chicago, Ill. 1909-13, N. Dakota School of Mines; E. M. 1910-12, summer, Illinois Steel Co., So. Chicago, Ill.

Present position: 1913 to date, Min. Engr., Ray Consolidated Copper Co.

Claude Freeman Lewis, Pottsville, Pa.

Proposed by H. M. Chance, G. B. Hadesty, Frank A. Hill, E. C. Luther, James Archbald, Jr.

Born 1870, St. Clair, Pa. 1889, Pottsville High School.

Present position: 1889 to date, Div. Supt., Philadelphia & Reading Coal & Iron Co.

Alexander D. Marriott, Jr., San Francisco, Cal.

Proposed by G. H. Clevenger, Robert H. Bradford, H. W. Young.

Born 1886, South Dakota. 1893-1904, General grammar and high school. 1904-06, Univ. of Nebraska, Lincoln, Neb.; M. E. 1907-09, Mogul Min. Co. 1910-13, Santa Gertrudis Min. Co. 1914, La Fe Min. Co., Zacatecas, Mex. 1914-15, Goldfield Cons. Min. & Mill. Co.

Present position: Representing Dorr Cyanide Machinery Co. at exhibit at Panama-Pacific International Exposition.

Alfred L. O'Brien, Anaconda, Mont.

Proposed by James K. Murphy, Louis V. Bender, Frederick Laist.

Born 1890, Boston, Mass. 1909, Grad., Chelsea High School, Chelsea, Mass. 1913, Lehigh Univ., So. Bethlehem, Pa.; E. M. in mining engineering and metallurgy. 15 months Assistant Testing Engr., Washoe Smelter, Anaconda, Mont.; two months Assay Laboratory.

Present position: Chemical Laboratory, Washoe Smelter.

Robert Matson Perry, Oak Creek, Colo.

Proposed by J. G. Perry, C. L. Colburn, Fred H. Bostwick.

Born 1884, Chicago, Ill. Grade schools of Denver, Colo. 1899-1902, Manual Training High School, Denver, Colo. 1902-03, Denver University. 1903-08, Columbia University; A. B. and E. M.

Present position: 1908 to date, Genl. Supt., Moffat Coal Co. and Oak Hills Coal Co.

Bruno Schettler, Benton, Ill.

Proposed by Frank A. Ray, Fred B. Nold, Trevor B. Simon.

Born 1877, Saxony, Germany. 1884-92, Common schools, Wellston, Ohio. 1901-04, Special Mine Engineering, Ohio State Univ. Prior to 1905 in coal mining generally. 1905-06, Min. Engr., Jones & Adams Co. and later Illinois Collieries Co. 1907-14, Supt., Peabody Coal Co.

Present position: 1914 to date, Genl. Mgr. and Sec.-Treas., Benton Coal Co.

Charles F. Schnepf, Cobalt, Ont., Canada.

Proposed by J. F. Kemp, Charles P. Berkey, Fred W. Young.

Born 1887, New York City. 1902, New York public schools. 1902-04, City College, New York. 1904-06, Columbia Grammar School, New York. 1908-10, Columbia University; E. M. 1907, summer, Transitman, Land Surveys, N. J. 1909, summer, Timberman, Copper Range Cons. Co., Painesdale, Mich. 1910, Mill Engr., St. Lawrence Pyrites Co., De Kalb, N. Y. 1910-12, Land Surveyor in N. J. Assistant Instructor Mine Surveying, Assistant in Metallurgy, Columbia University, N. Y. 1912-13, Field Engr., A. S. & R. Co., Nickel Exploration, Sudbury, Ont. 1913, Underground Foreman, Creighton Mine, Ont. 1913, Min. Geol., Canadian Copper Co., Copper Cliff, Ont. 1914-15, Min. Engr., Assayer, Millman, Nipissing Mining Co., Cobalt, Ont.

Present position: Min. Engr. and Assayer, Right of Way Mining Co.; Min. Engr., Meteor Mining Co.

Frank Edward Shepard, Denver, Colo.

Proposed by F. K. Brunton, Bradley Stoughton, D. W. Brunton.

Born 1865, Ashland, N. H. 1883, Grad., Dorchester High School. 1887, Mass. Inst. of Technology; M. E. 1887, Machinist, B. & A. R. R., Boston. 1888, Asst. Inspector of Steam Vessels, Boston. 1890, Mech. Engr., Thomas, Shepard & Searing, Denver, Colo. 1891, Shepard & Searing, Mech. and Elec. Engrs. 1895, M. E., Denver Engineering Works.

Present position: 1897 to date, Pres., Denver Engineering Works.

Armand Tissot, Paris, France.

Proposed by W. L. Saunders, George A. Howells, Henry Lang.

Born 1861, Bourgnignon. Ingenieur des Arts et Metiers. 1878-81, Chalons sur Marne.

Present position: 1902 to date, Genl. Representative Ingersoll-Rand Co. in the north of France.

Greyson Prevost Troutman, Jeddo, Pa.

Proposed by John Markle, A. B. Jessup, H. Eckfeldt.

Born 1880, Centralia, Pa. 1895, Grad., Centralia High School. 1896, Woods' Business College. 1906, Bethlehem Preparatory School. 1906-10, Grad., Lehigh University; M. E. 1897-1900, Engrg. Dept., Lehigh Valley Coal Co. 1900-02, District Transitman, M. and S. Div., Lehigh Valley Coal Co. 1902-05, Asst. Div. Engr., M. and S. Div., Lehigh Valley Coal Co. During vacations worked for Lehigh Valley Coal Co. and Westmoreland Coal Co. 1910-11, Div. Engr., Delaware Div., Lehigh Valley Coal Co. 1911, Div. Engr., Lackawanna Div., Lehigh Valley Coal Co. 1912, Asst. Div. Supt., Lackawanna Div., Lehigh Valley Coal Co.

Present position: Asst. Genl. Mgr., G. B. Markle Co.

E. O. Wilson, Los Angeles, Cal.

Proposed by B. K. Stroud, Thomas Cox, A. K. P. Harmon, Jr.

Born 1870, Darke Co., Ohio. Public schools of Los Angeles. One year in Stanford University. Designer and manufacturer of oil-well tools. 1897-1904, Baker Iron Works, during which time I invented the Wilson patented casing spear, also Wilson under-reamer. 1904-09, Mgr., Bakersfield Iron Works. Inventor of Wilson casing elevators.

Present position: 1909 to date, Pres., Wilson & Willard Mfg. Co.

William James Wolf, Worthington, W. Va.

Proposed by A. C. Beeson, C. E. Krebs, J. M. Clark.

Born 1884, Shamokin, Pa. 1902, Grad., Shamokin High School. 1904-05, Studied but did not complete course of Mining Engineering in I. C. S. 1903, Eight months, Engr. Corps, Susquehanna Coal Co., Lykens, Pa. 1903-06, Draftsman, Monongahela River Consolidated Coal & Coke Co. 1906-13, Draftsman and Resident Engr., Four States Coal & Coke Co., Dorothy, W. Va. 1913-14, Engineer Dept., Pittsburgh-Buffalo Co. and Four States Coal & Coke Co., Canonsburg, Pa.

Present position: Resident Mgr. Annabelle Mine for Receivers Four States Coal & Coke Co.

Harry F. Wright, Tulsa, Okla.

Proposed by William J. Linn, S. W. Beyer, H. H. Stoeck.

Born 1888, Madison, Wis. 1906, High School, Carroll, Ia. 1908-12, Iowa State College; B. S. 1912-13, Advanced work in geology and part time instructor in geology, Iowa State College. 1911, summer, U. S. G. S., Chief of party, M. A. Fishel. 1913-14, Iowa Engineering Experiment Station. Geologic work on Iowa road materials with Dr. S. W. Beyer. Report published by Iowa Geological Survey.

Present position: 1914 to date, Geol., Gypsy Oil Co.

Rufus Johnston Wysor, So. Bethlehem, Pa.

Proposed by H. A. Brassert, E. O'C. Acker, W. L. Cumings.

Born 1885, Dublin, Va. 1906, Virginia Polytechnic Institute; B. S. in Applied Chemistry and Metallurgy. 1906, Chem., Virginia Iron, Coal & Coke Co. 1906, Asst. Chem., Duquesne Steel Works, Carnegie Steel Co. 1908, First Asst. Chem., Carnegie Steel Co. 1910, Member of firm, Kann & Wysor. 1911, Metallurgical Inspector, Panama Canal Commission, Emergency Dams. 1912, Chief Chem., Bethlehem Steel Co.

Present position: 1913 to date, Chief Chem. and Engr. of Tests, Bethlehem Steel Co.

William LeVerne Zeigler, Sunset, Idaho.

Proposed by Rush J. White, C. Eustace Dwyer, E. Percy Smith.

Born 1890, Moscow, Idaho. 1905, High School. 1910, Univ. of Idaho; B. S. in Min. Engrg. 1907, Miner, Morning Mine. 1910, Millman, Hewitt Mine. 1911, Supt. of Mill, Monarch Mine, British Columbia.

Present position: 1913 to date, Mill Supt., Success Mining Co.

Associate Member

James Ferguson Burns, Colorado Springs, Colo.

Proposed by E. P. Arthur, Jr., John T. Hawkins, J. D. Hawkins.

1891-1905, Pres. and Genl. Mgr., Portland Gold Mining Co., Cripple Creek, Colo. 1905 to date, Pres. and Mgr. of other mines in Colorado and Nevada.

Present position: 1914 to date, Pres., Cripple Creek Deep Drainage & Transportation Tunnel Co.

Junior Members

Glenn Lee Allen, Lawrence, Kan.

Proposed by E. S. Dickinson, C. M. Young, C. S. Heislar.

Born 1889, Saffordville, Kan. 1909, Emporia High School, Emporia, Kan. 1911, Surveyor, Uncle Sam Oil Co. 1914, Chem., Copper Queen Consolidated Mining Co.

Present position: Student, University of Kansas.

Hugh Reid Brown, Lawrence, Kan.

Proposed by E. S. Dickinson, C. M. Young, E. C. Reeder.

Born 1888, Fleming, Mo. Pleasanton High School. 1910, Kansas State Agricultural College. 1911, Kansas Univ., now Senior in Mining Department. 1909-10, Six months, helper to Mining Engineer, M., K. & T. Coal Dept. 1912-13, Two summers, M., K. & T. at Mine No. 12, Coalgate, Okla. 1904-05, Central Coal & Coke Co., Scammon, Kan.

Present position: Student, Kansas University.

Alfred C. Levis, Golden, Colo.

Proposed by Harry J. Wolf, William R. Chedsey, F. W. Traphagen.

Born 1894, Baltimore, Md. 1912, Baltimore Polytechnic Institute. Have just completed the Junior year at the Colorado School of Mines.

Present position: Student, Colorado School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period June 10 to July 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 103, April to July, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of July 10, 1915.

ADDICKS, LAWRENCE	Room 2, 126 Liberty St., New York, N. Y.
ALINE, PETER A.	1525 Eighth Ave. N., Great Falls, Mont.
ALLEN, ARTHUR POTTER, Engr.	Winona Copper Co., Winona, Mich.
ARENTZ, SAMUEL S.	303 Dooly Bldg., Salt Lake City, Utah.
ASHMORE, ERNEST P.	504 Sherbourne St., Toronto, Ont., Canada.
BATCHELLER, J. H., Supt., U. S. Smelting, Refining & Mining Explor. Co.,	Joplin, Mo.
BAUMANN, HENRY N., JR.	Chicago, Alaska.
BINGHAM, RAYMOND A.	Care American Zinc & Chemical Co., Langeloth, Pa.
BIRCH, STEPHEN	120 Broadway, New York, N. Y.
BONSBIE, ROY SAMUEL	Care Low Moor Iron Co., Low Moor, Va.
BROWN, ALBERT L., Wallaroo & Moonta Min. & Smelt. Co., Ltd., Wallaroo,	So. Australia.
CAHEN, JAMES P., JR.	353 Central Park West, New York, N. Y.
CALLAWAY, L. A.	Anaconda, Mont.
CARLETON, JAMES G.	Marblehead, Mass.
CARLSON, A. E.	Magpie Mine, Ontario, Canada.
CARPENTER, A. B.	508 Union League Bldg., Los Angeles, Cal.
CARPENTER, ARTHUR H.	210 Noble Ave., Grafton, Pa.
CARTLIDGE, OSCAR	Gillespie, Ill.
CHASE, F. D.	Dedham Ave., Dedham, Mass.
CHURCH, JOHN L.	Instructed to hold all mail.
CLEMES, A. WILLIS	Instructed to hold all mail.
COCHRAN, RALF S.	Met. Republic Iron & Steel Co., Youngstown, Ohio.
CORBIN, J. ROSS	American P. O. Box 926, Shanghai, China.
CORWIN, FRANK R.	759 12th St., Douglas, Ariz.
CROMWELL, ROBERT H., Compania Metalurgica Mexicana, 82 Beaver St.,	New York, N. Y.
DAVIS, CARL R., Genl. Mgr., Brakpan Mines, Ltd., Box 3, Brakpan, Transvaal,	S. Africa.
DAVISON, G. L.	525 Oak St., Chattanooga, Tenn.
DODGE, W. E.	301 S. Guthrie Ave., Tulsa, Okla.
EDWARDS, JOHN R.	Navy Dept., Washington, D. C.
EMERY, A. B., Gen. Mgr., Messina Transvaal Development Co., Ltd., Messina,	Transvaal, S. Africa.
EVANS, CADWALLADER, JR.	1440 Henry W. Oliver Bldg., Pittsburgh, Pa.
FERNAN, J. J. C.	47 Lanercost Road, Tulse Hill, London, S. W., England.
FOGG, LEWIS WARNER, JR.	Care Mrs. Barclay, 107 McKean Ave., Donora, Pa.
FOX, WALTER V.	Care Alvarado Mining Co., Parral, Chih., Mexico.
FRASER, COLIN	Care Bank of New Zealand, Sydney, N. S. W., Australia.
FREYMUTH, WILLIAM A.	Maihar, E. I. Ry., India.

- FULLER, JOHN T. Box 393, Little Rock, Ark.
 FURNESS, DWIGHT 614 Mills Bldg., El Paso, Tex.
 GAEBELEIN, PAUL W. Mill Supt., Baker Mines Co., Cornucopia, Ore.
 GOODRICH, CHARLES CROSS 60 Broadway, New York, N. Y.
 GOODRICH, R. R. Stanhope, N. J.
 GORMAN, THOMAS C. Creighton Mine, via Sudbury, Ont., Canada.
 HANCE, JAMES H. 202 Perry St., Elgin, Ill.
 HANLON, JOHN EDWARD Timmins, Ont., Canada.
 HEISLAR, CLARENCE S. Lowell, Ariz.
 HENRY, WILBUR E. 927 Old National Bank Bldg., Spokane, Wash.
 HERZIG, C. S. 48 W. 25th St., New York, N. Y.
 HONNOLD, WILLIAM L. 147 Fraser Ave., Ocean Park, Cal.
 HUNTER, CHARLES, Gen. Mgr., Socorro Gold & Silver Mine, Ltd.,
 Valle de Angeles, Tegucigalpa, Honduras.
 IMHOFF, WALLACE G. Corrigan, McKinney & Co., Cleveland, Ohio.
 JONES, WILLIAM Linden Avenue, Gosforth, Newcastle-on-Tyne, England.
 KENNEDY, JOSEPH E. 120 Broadway, New York, N. Y.
 KERR, DAVID G. Carnegie Bldg., Pittsburgh, Pa.
 KING, CHARLES F. 49 Prospect St., Brooklyn, N. Y.
 KLOPSTOCK, PAUL 15 Broad Street, New York, N. Y.
 KNUTZEN, THEODOR, Care Boston Chemical Co., 112 N. Plum St., Richmond, Va.
 KUANG, Y. C., Care Canton Christian College, 156 Fifth Ave., New York, N. Y.
 KUNDSEN, BJARNE Algoma, Wis.
 LAKE, MACK CLAYTON 119 E. Mifflin St., Madison, Wis.
 LARSON, CLARENCE LEO Kellogg, Idaho.
 LEE, EDWARD CLARENCE 58 Broad St., Stroudsburg, Pa.
 LINDBERG, CARL O. 506 Union League Bldg., Los Angeles, Cal.
 MCKENNA, ALEXANDER G., 621 Alabama Ave., S.E., Congress Heights,
 Washington, D. C.
 MCKILLOP, JOHN, Care W. Muller, 69a Great Queen Street, Kingsway,
 London, W. C., England.
 MCLEOD, ALEXANDER C. S., 2 Wright St., Middle Park, Melbourne, Vic., Australia.
 MCMEEKIN, CHARLES W., Surface Supt., Goldfield Consolidated Mines Co.,
 Goldfield, Nev.
 MACGEORGE, ALEXANDER J., Care Athenæum Club, Collins St., Melbourne, Vic.,
 Australia.
 MAIN, ALFRED F., Care Exploration Co., Ltd., 61 Broadway, New York, N. Y.
 MANN, WILLIAM SEWARD 236 Douglas Bldg., Los Angeles, Cal.
 MEEKS, R. V. Care Vinton Colliery Co., 1 Broadway, New York, N. Y.
 MILLER, CHARLES LEO, Chief Engr., Scottsdale Machine & Mfg. Co., Scottsdale, Pa.
 MILLS, FRANK P. 1801 Francisco St., Berkeley, Cal.
 MOORE, EDWARD W., Care J. F. Gilchrist, 5343 Dorchester Ave., Chicago, Ill.
 MUNROE, HENRY S. Litchfield, Conn.
 OLIVER, ROLAND B. 3834 Beaumont Ave., Oakland, Cal.
 ORR, CHESTER A. Mgr., Cleveland Metal Products Co., Cleveland, Ohio.
 ORROK, GEORGE A. Lee, Mass.
 PALMER, IRVING A. Supt., United States Smelting Co., Altona, Kan.
 PERRY, OSCAR B., Min. Engr., Gen. Mgr., Yukon Gold Co., 120 Broadway,
 New York, N. Y.
 PITMAN, S. M. Box 1118, Providence, R. I.
 POTTER, EDWARD C. Clearwater, Missoula Co., Mont.
 PROCHAZKA, GEORGE A., JR. . . . Central Dye Stuff Chemical Co., Newark, N. J.
 RADCLIFFE, DONALD HEWSON, Geol. Gypsy Oil Co., Tulsa, Okla.
 REEDER, EDWIN C. 401 W. State St., Marshalltown, Ia.
 RICHARDSON, W. V. 6451 Hillegass St., Oakland, Cal.
 RODGERS, JOSEPH H. R. F. D. 10, Box 291, Los Angeles, Cal.
 RODGERS, SELDEN S. Care Union Basin Mining Co., Goconda, Ariz.
 RORE, FRANK C. Care A. E. Palmer, Rapid City, Mich.
 RUEBEL, ERNST HERTEL, JR. General Delivery, Collinsville, Okla.
 SCHEUCH, WILLIAM ALLEN 302 Hull Ave., New York, N. Y.
 SCHLEICHER, HENRY M., Ore Dressing 62 Forest St., Boston, Mass.
 SHERMAN, SCOTT H., Supt., Christmas Mine, American Smelting & Refining Co.
 Christmas, Ariz.
 SIEBERT, FREDERIC J. 2526 Broadway, San Francisco, Cal.
 SMITH, LLOYD B. Tulsa, Okla.
 STOCKETT, ALFRED W. Mauch Chunk, Pa.

STRAMLER, ALLEN P.	Pinos Altos, N. M.
SWEENTY, HARRY P.	32 N. Massachusetts Ave., Atlantic City, N. J.
TAYLOR, WILLIAM H.	Care Librarian, Phillips Academy, Andover, Mass.
TIMMONS, COLIN	Supt., South Carolina Mine, Angels Camp, Cal.
TRAUERMAN, CARL J., Min. Engr.,	Care Rothwell-Trauerman, 832 Colorado St., Butte, Mont.
TREMOUREAUX, ROY E., Asst. Supt.,	North Star Mines, Champion Mines, Nevada City, Cal.
TURNER, G. D. B., Care W. K. Chandler & Sons,	4 Lloyds Ave., London, E. C., England.
TURNER, THOMAS NORTON	Dawson, Y. T., Canada.
VAN DORP, G. H.	Care Liberty Bell Mine, Telluride, Colo.
VENABLES, HARRY L.	Casilla 105, Oruro, Bolivia.
VRANG, CHRISTIAN M.	Barkerville, Cariboo Dist., B. C., Canada.
WADDELL, GEORGE FULTON	Squirrel, Fremont Co., Idaho.
WALTERS, MAURICE B.	Instructed to hold all mail.
WANVIG, JOHN D., JR.	Care Union Basin Mining Co., Golconda, Ariz.
WHITE, R. T.	Care F. A. Mattievitch & Co., Batoum, Russia.
WILBRAHAM, A. G. B., Care W. O. D. Greene,	87 Cannon St., London, E. C., England.
WILLIAMS, J. C.	Morenci, Ariz.
WILLIAMS, W. A., Chief Petroleum Engr.,	Bureau of Mines Experimental Station, 506 Custom House, San Francisco, Cal.
WILLIAMSON, H. D.	83 Hayes Road, Bromley, Kent, England.
WILLIS, BAILEY	Geological Department, Stanford, Cal.
WOLF, ALBERT G.	Box 543, Golden, Colo.
WRAITH, CHARLES R., Anaconda Copper Mining Co.,	Exhibit, Palace of Mines, P. P. I. E., San Francisco, Cal.
WRAITH, WILLIAM, Genl. Mgr., International Smelting Co.,	622 Kearns Bldg., Salt Lake City, Utah.
YOUNG, RALPH WILLIAM	Hammond Laboratory, New Haven, Conn.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
BOISE, CHARLES WATSON,	Care Forminière, 8 Montagne du Parc, Brussels, Belgium.
CLERC, CAMILLE	92 Rue Jouffrey, Paris, France.
DOWLER, HARRY P.	Penn-Mary Coal Co., Heilwood, Pa.
EATON, E. R.	Manatawny Bessemer Ore Co., Manatawny, Pa.
HEBBARD, AUSTIN	Box 98, Langlaate, Transvaal, South Africa.
JONES, THOMAS J.	Perm Govt., via Petrograd, Russia.
KIRKLAND, T. C., Cia. Met. y Refinadora del Pacifico,	S. A., Fundicion, Son., Mexico.
KNOERTZER, H.	56 Rue Balagny, Paris, France.
MCCARRICK, E.	628 N. Serrano Ave., Los Angeles, Cal.
MORRIS, CHARLES E.	Pony, Madison Co., Mont.
MORSE, ROBERT G.	Mass. Steel Casting Co., Everett, Mass.
OLIVER, ROLAND B.	8 Montagne du Parc, Brussels, Belgium.
ROCHLING, H.	Völklingen Bei Saar, Rhenish Prussia, Germany.
RUBEL, MILTON L., Pilonas Min. Co., Topia, Dgo., Mex.,	via Nogales and Culiacan, E. De Sin.
WILCOX, RALPH	Curanilahue, Chile.
WRIGHT, JESSE T.	Wallace, Idaho.

NECROLOGY

The death of the following member was reported to the Secretary's office during the period June 10 to July 10, 1915:

Date of Election.	Name.	Date of Decease.
1895	* Edward Skewes	May 23, 1915

* Member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*, JOHN H. JANEWAY, *Vice-Chairman*.
 F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
 P. A. MOSMAN, *Treasurer*.

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*, ALFRED C. LANE, *Vice-Chairman*.
 HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
 ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.

FRANK A. ROSS, *Chairman*, RUSH J. WHITE, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.

I. F. LAUCKS, *Chairman*, J. F. MENZIES, *Vice-Chairman*.
 GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
 H. L. MANLEY.

Southern California

SEELEY W. MUDD, *Chairman*, C. COLCOCK JONES, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
 RALPH ARNOLD, A. B. CARPENTER.

Colorado

CHARLES A. CHASE, *Chairman*, S. A. IONIDES, *Vice-Chairman*.
 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*, JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.

G. HOWELL CLEVINGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
 JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
 F. W. BRADLEY, ANDREW C. LAWSON.

*Pennsylvania Anthracite Section*R. V. NORRIS, *Chairman*.

CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
 EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
 CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

ARTHUR THACHER, *Chairman*, R. A. BULL, *Vice-Chairman*.
 WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 H. A. BUEHLER, R. R. S. PARSONS, HERBERT A. WHEELER.

Chicago

ROBERT W. HUNT, *Chairman*, J. A. EDE, *Vice-Chairman*.
 HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.
 F. K. COPELAND, G. M. DAVIDSON.

Utah

R. C. GEMMELL, *Chairman*, C. W. WHITLEY, *Vice-Chairman*.
 ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
 WALTER FITCH, WILLIAM WRAITH.

STANDING COMMITTEES

*Executive*WILLIAM L. SAUNDERS, *Chairman*GEORGE D. BARRON,
SIDNEY J. JENNINGS,JOSEPH W. RICHARDS,
BENJAMIN B. THAYER.*Membership*JOHN H. JANEWAY, *Chairman*KARL EILERS,
LEWIS W. FRANCIS,LOUIS D. HUNTOON,
ARTHUR L. WALKER.*Finance*GEORGE D. BARRON, *Chairman*.

ALBERT R. LEDOUX,

CHARLES F. RAND.

*Library*E. GYBBON SPILSBURY, *Chairman*.¹KARL EILERS²
JOHN HAYS HAMMOND,¹ALEX C. HUMPHREYS,²
BRADLEY STOUGHTON.*Papers and Publications*BRADLEY STOUGHTON, *Chairman*.

EXECUTIVE COMMITTEE

KARL EILERS,
JOSEPH W. RICHARDS,GEORGE C. STONE.
SAMUEL A. TAYLOR.J. L. W. BIRKINBINE,
WILLIAM H. BLAUVELT
H. A. BRASSERT,
WILLIAM CAMPBELL,
R. M. CATLIN,
ALLAN J. CLARK,
FREDERICK G. COTTRELL,
NATHANIEL H. EMMONS,
JOHN W. FINCH,
CHARLES H. FULTON
F. LYNWOOD GARRISON,
ROBERT C. GEMMELL,
CHARLES W. GOODALE,
HARRY A. GUESS,
R. DAWSON HALL,
PHILIP W. HENRY,
HEINRICH O. HOFMAN,WALTER E. HOPPER,
HENRY M. HOWE,
LOUIS D. HUNTOON,
J. E. JOHNSON, JR.,
LEE O. KELLOGG,
WILLIAM KELLY,
JAMES F. KEMP,
CHARLES K. LEITH,
ANTHONY F. LUCAS,
EDWARD P. MATHEWSON,
HERBERT A. MEGRAW,
RICHARD MOLDENKE,
SEELEY W. MUDD,
R. V. NORRIS,
EDWARD W. PARKER,
EDWARD D. PETERS,
ROSSITER W. RAYMOND,THOMAS T. READ,
ROBERT H. RICHARDS,
L. D. RICKETTS,
HEINRICH RIES,
E. F. ROEBER,
RENO H. SALES,
ALBERT SAUVEUR,
HENRY L. SMYTH,
A. A. STEVENSON
RALPH H. SWEETSER,
FELIX A. VOGEL,
ARTHUR L. WALKER,
ROLLA B. WATSON,
HORACE V. WINCHELL,
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Roasting and Leaching Concentrator Slimes Tailings

BY LAWRENCE ADDICKS, DOUGLAS, ARIZ.

(San Francisco Meeting, September, 1915)

THROUGH the courtesy of Dr. James Douglas I am permitted to give a summary of some of the results obtained in leaching slimes tailings in a series of experiments conducted under my direction at Douglas, Ariz., during the past year.

While the test run for which the data are given deals with a particular case, much of the work is general in its application and may therefore be of interest.

This work was done upon the ores of the Burro Mountain Copper Co. A description of the geology of the district will be found in the paper by R. E. Somers published in the May, 1915, *Bulletin* of the Institute.

The mineral is pyrite and chalcocite very finely disseminated throughout the gangue and a characteristic of all the ores is the large amount of alumina present.

A representative analysis of the particular ore under consideration would be about as follows:

Cu, per cent.....	2.35	S, per cent.....	3.0
Ag, ounces per ton.....	0.05	CaO, per cent.....	0.5
Au.....	None	K ₂ O, per cent.....	4.0
Fe, per cent.....	1.5	SiO ₂ , per cent.....	70.0
Al ₂ O ₃ , per cent.....	13.0		

The leaching of the ore direct was early dismissed from consideration for the following reasons:

1. The ore would have to be quite finely crushed to liberate the mineral for leaching; concentrating after crushing would cost but a few cents a ton; and a higher total recovery would result from leaching tailings rather than ore.

2. Rough concentration would yield a suitable material for roasting to make sulphur dioxide or sulphuric acid in connection with the leaching scheme.

3. Such a program put the question of cost vs. recovery on a basis where flotation and discarding of tailings were directly comparable.

Concentrating experiments already available indicated that the tail-

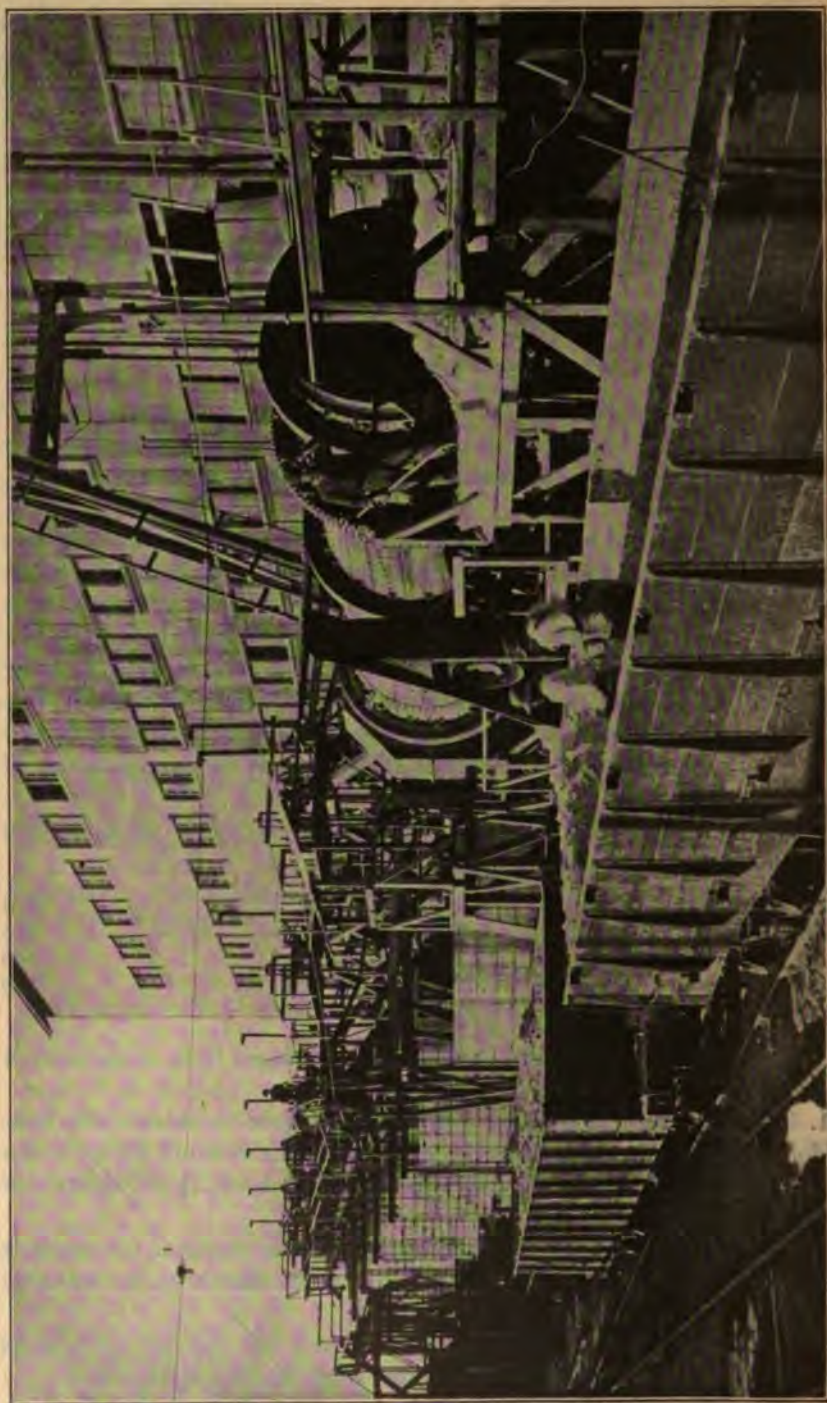


FIG. 1.—GENERAL VIEW OF LEACHING PLANT.

ings would consist of about 50 per cent. of sands running about 0.3 per cent. in copper, and therefore too low to justify re-treatment, and 50 per cent. of slimes running perhaps 1 per cent. in copper, and it was on this latter material that the large-scale work was finally done.

At first flotation gave very unsatisfactory results, due primarily to the readiness with which the fine mineral particles became coated with a film of oxide upon exposure while concentrating, but later developments in this field overcame these difficulties and it became apparent just about the time the leaching plant was being started that the concentrator recovery was going to be so high that re-treatment of the tailings would not be justified. Nevertheless, for the sake of the data it was decided to carry through a representative run, the results of which are discussed below.

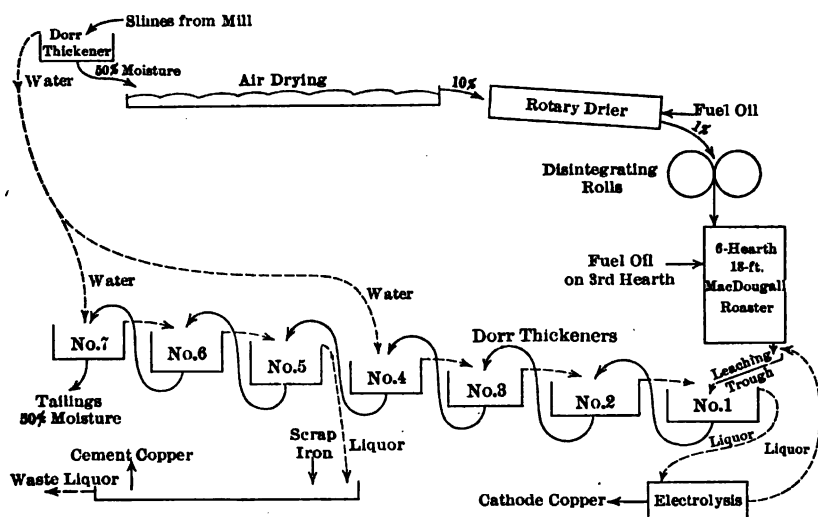


FIG. 2.—FLOW SHEET OF LEACHING PLANT.

Fig. 1 is a view of the leaching plant, and the general scheme is shown in flow sheet, Fig. 2. As the development of the electrolytic precipitation is really a subject apart from the leaching of the slimes, the results from this part of the work have been reserved for a separate paper. A part of the copper is shown recovered as cement in order to give a means of holding the iron and alumina in the liquor to be electrolyzed down to the desired point. A high ratio of washing can be used in the thickeners delivering to the cementation launder; in the remaining thickeners the ratio is governed by the evaporation loss, as there is no other outlet for liquor.

The experiments will be taken up in detail under the headings of: (1) drying, (2) roasting, (3) leaching, and (4) washing.

Drying

As the slimes would be delivered from the concentrator in suspension as a thin pulp it became necessary to consider carefully the question of drying.

Thickening experiments indicated that there was a critical degree of moisture of about 45 per cent., below which pumping the settled slimes resulted in further classification of the fine particles instead of delivering

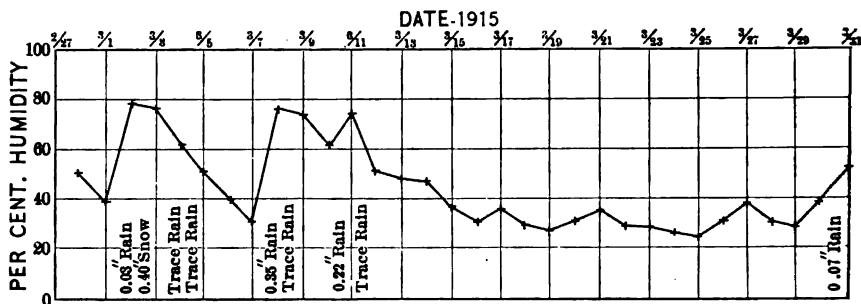


FIG. 3.—MOISTURE IN AIR.

average pulp, so that thickening in a standard Dorr unit to 50 per cent. of moisture was established as the first step.

As fuel is very expensive at the location in question and no waste heat can be counted on except that from the roaster gases, experiments were made with drying by exposure to the desert sun and winds. The heavy pulp could be poured into a shallow basin and the more or less dried slime reclaimed by some mechanical method for much less expense than

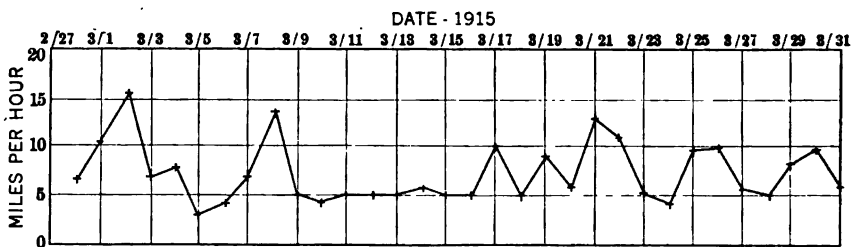


FIG. 4.—WIND VELOCITY.

an equivalent drying using artificial heat would cost. It was found that the pulp settled to about 30 per cent. of moisture very readily when allowed to drain and that reasonably dry material could be obtained by a month's standing.

Figs. 3, 4, 5, and 6 give the measurements taken during the trial on three test beds, 10 ft. square and respectively 2, 6, and 10 in. deep, which were exposed to the atmosphere for about a month.

The climatic conditions are shown in Figs. 3, 4, and 6 and it will be noted that periods of both cold and wet weather were encountered during the first part of the run, much more favorable conditions following in the last two weeks. A bed only 2 in. deep would cost a good deal to reclaim, but the rate of drying, as shown in Fig. 5, was thought to indicate that a

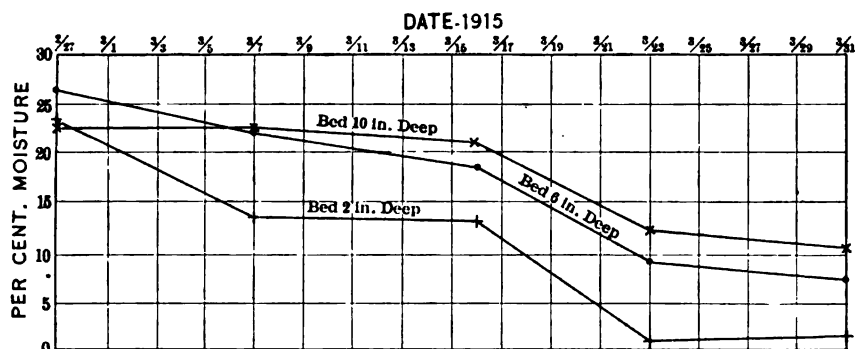


FIG. 5.—MOISTURE IN SLIMES.

reasonably thick bed could be brought down to 10 per cent. of moisture without an excessive period of standing, and it seems probable that this will be the cheapest way to obtain this preliminary drying.

Up to this point the experiments were conducted on relatively small quantities of material, but starting with the following step—drying in the rotary kiln—large tonnages were handled. Instead of using atmospheric

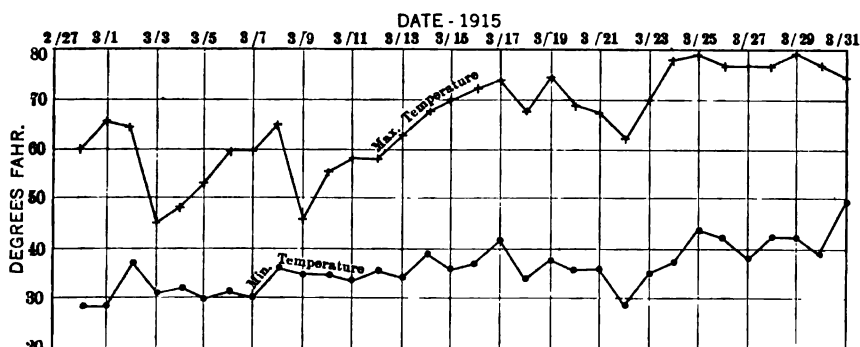


FIG. 6.—TEMPERATURE OF AIR.

drying before feeding to the kiln, steam coils were buried in the cars of mushy slimes shipped to the experimental plant, bringing the moisture down to about 25 per cent., when it was shoveled directly to the drier.

This drier was one of the old Douglas roasters, 7 ft. in diameter and 31 ft. long. It was used as an ordinary unlined rotary kiln, with channel-

iron spilling blades throughout the first two-thirds of its length and cast-iron balls 3 in. in diameter feeding against a delivery screen made of sheet-iron plate with $1\frac{1}{2}$ -in. round holes. The idea was to dry and pulverize any caked nodules in the one apparatus. The outside was lagged with $1\frac{1}{2}$ -in. magnesia block and heat was supplied by three oil burners at the delivery end. Two typical runs gave the following results:

Days run.....	23.0	12.0
Total tons dried.....	710.0	388.0
Tons per day.....	30.9	32.3
Per cent. moisture entering.....	26.6	10.0
Per cent. moisture leaving.....	1.1	2.0
Gallons fuel oil per day.....	404.0	154.0
Gallons fuel oil per ton dried.....	13.1	4.8

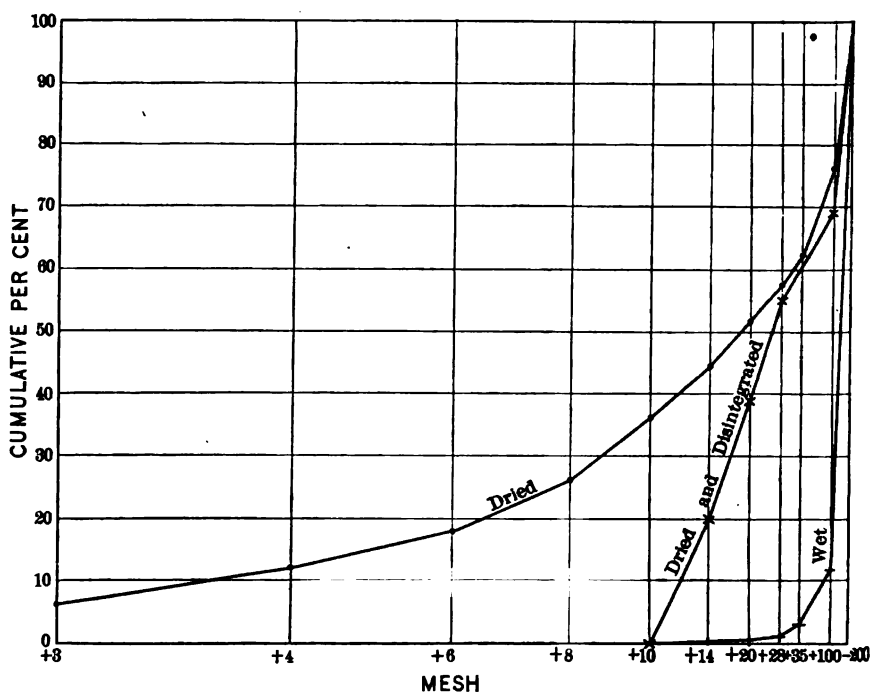


FIG. 7.—CUMULATIVE SCREEN ANALYSIS OF SLIMES.

In general, it was concluded that about $\frac{1}{2}$ gal. of fuel oil must be provided for each unit of moisture in the ore to be dried, so that 5 gal. would have to be charged against drying a 10 per cent. product from the atmospheric drying beds, although this amount might be lowered in a properly designed drier. The capacity of the drier was too low for the roaster used and the pulverizing action of the iron balls inadequate, so the dried material was passed through a pair of disintegrating rolls, returning any

oversize through a 10-mesh screen, and dried and disintegrated material was stored ahead of the roaster.

Fig. 7 shows cumulative screen analyses of the original wet slimes, the drier product, and the disintegrated material, made on the usual Tyler direct diagram. It must be remembered that the lumps formed in the kiln could be readily crushed between the thumb and forefinger, being merely caked clay. The danger was that these lumps would be burned hard in the subsequent roasting.

The arrangement of the drier with skip feeding to roaster, and the row of Dorr thickeners, can be readily seen in Fig. 1.

Roasting

The roasting was carried out in a standard six-hearth, 18-ft. Mac-Dougall furnace with water-cooled arms, fuel oil being introduced on the third hearth. No change was made in the furnace for this work except to

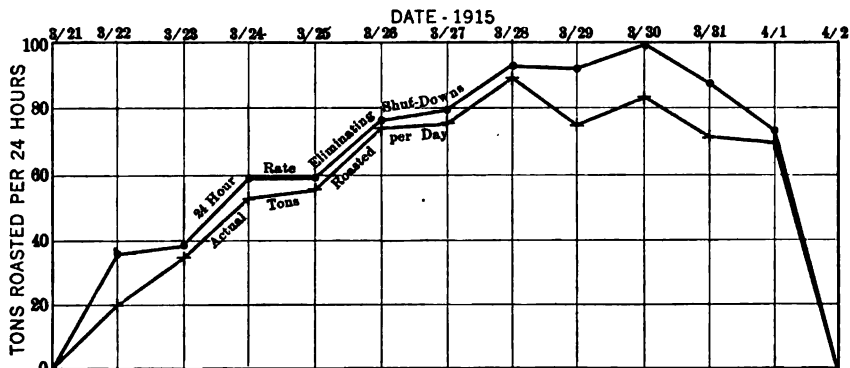


FIG. 8.—TONS ORE TO ROASTER.

replace parts of the upper brickwork with firebrick. Fig. 8 shows the tonnage fed to the furnace daily during the particular run with which the remainder of this paper chiefly deals. When the arms began to be buried the speed of rotation was increased from one revolution in 69 sec. to one in 60 sec. No trouble was experienced in the operation of the furnace.

Fig. 9 shows how the fuel-oil consumption was gradually cut down as the size of charge was increased. It is evident that 100 tons a day can be roasted with about 3.5 gal. of fuel oil per ton. The fuel oil burned was regulated so as to keep the maximum temperature on any hearth at about 950° F. Previous small-scale experiments had indicated that the proper roasting range lay between 900° and 1,100° F. Fig. 10 shows a log of the daily temperatures on the second, third, fourth, and fifth hearths. It will be noted that the heat zone descended in the furnace as the tonnage increased. This is shown diagrammatically in Fig. 11. At the same time

the efficiency of the roast increased, as shown by Fig. 12. This is apparently due to the fact that at the higher rates roasting temperatures were obtained on the fourth and fifth hearths, which were free from the more or less reducing atmosphere of the third and higher hearths due to the fuel oil.

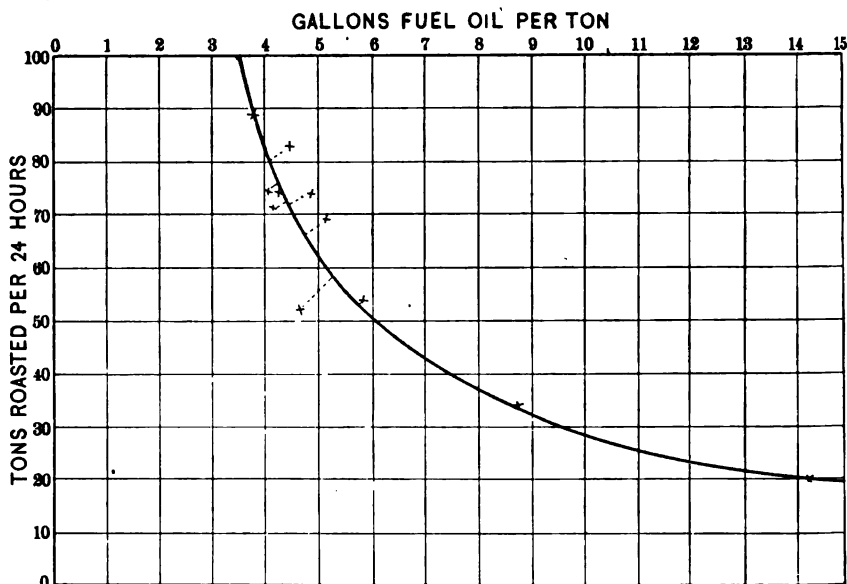


FIG. 9.—ROASTER TONNAGE VS. FUEL-OIL CONSUMPTION.

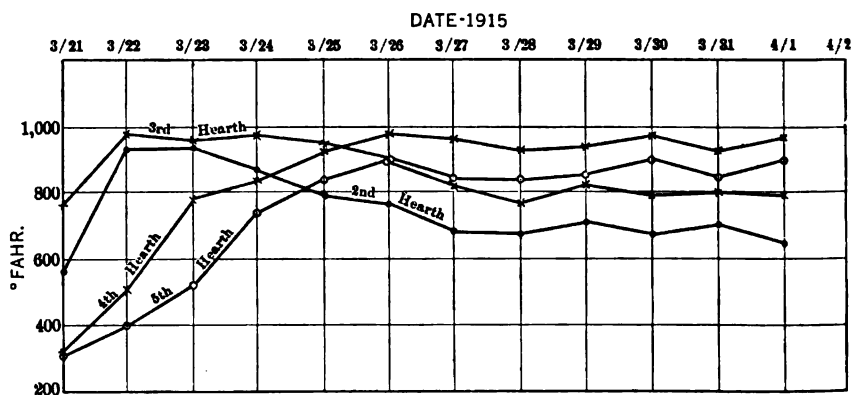


FIG. 10.—ROASTER HEARTH TEMPERATURES.

Leaching

The hot calcines from the roaster were delivered through a spout casting made of Duriron into a triangular trough about 30 ft. long sloping $1\frac{1}{4}$ in. to the foot, which delivered directly into the first of the series of

Dorr thickeners used for washing. This trough served three purposes: (1) that of leaching by agitation; (2) that of mechanically conveying the calcines from the roaster to the washing apparatus; and (3) that of absorbing the sensible heat of the calcines, thereby furnishing a warm liquor

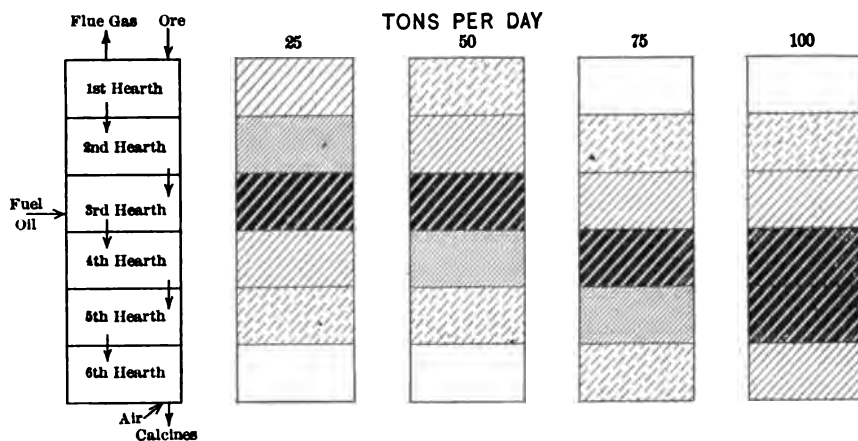


FIG. 11.—ROASTER HAERTH TEMPERATURES.

for electrolysis. A similar arrangement is described as doing excellent work in another field by Hofmann in his *Hydrometallurgy of Silver*. A deep triangular trough was first constructed of common lumber. This was then lined with sheet lead to make it liquor proof. Inside the lead

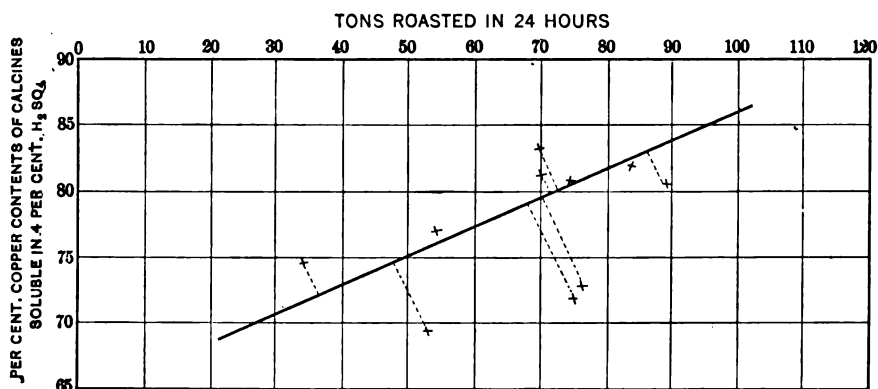


FIG. 12.—VARIATION OF ROASTER EFFICIENCY WITH TONNAGE.

lining, boards were loosely laid to prevent mechanical erosion of the lead by the calcines. A flat wooden cover was laid on top of the launder to keep down the steam and dust. This latter was not excessive, and would have been very slight had a more even delivery of calcines been made at

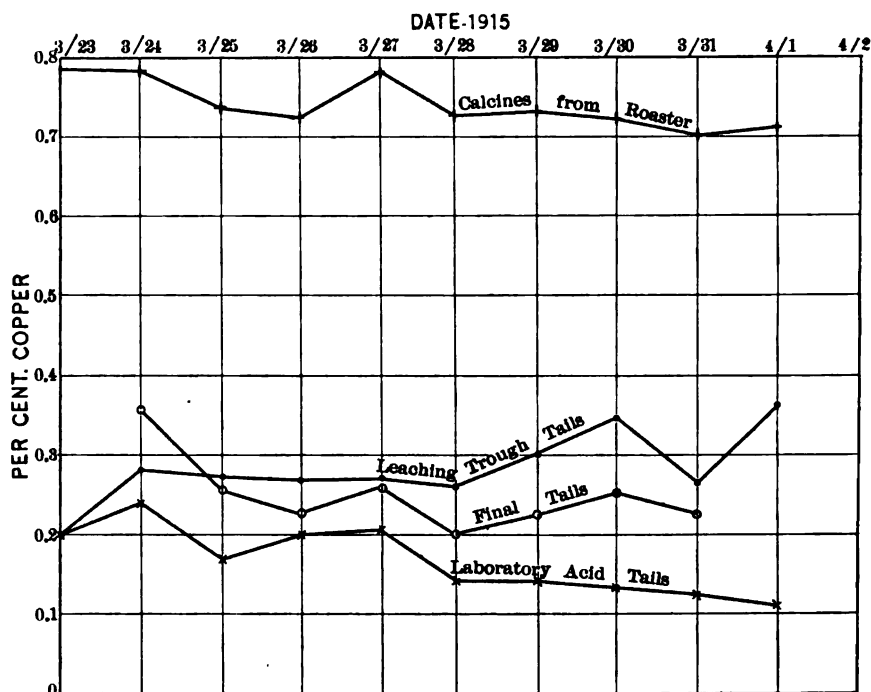


FIG. 13.—COPPER CONTENTS OF TAILINGS.

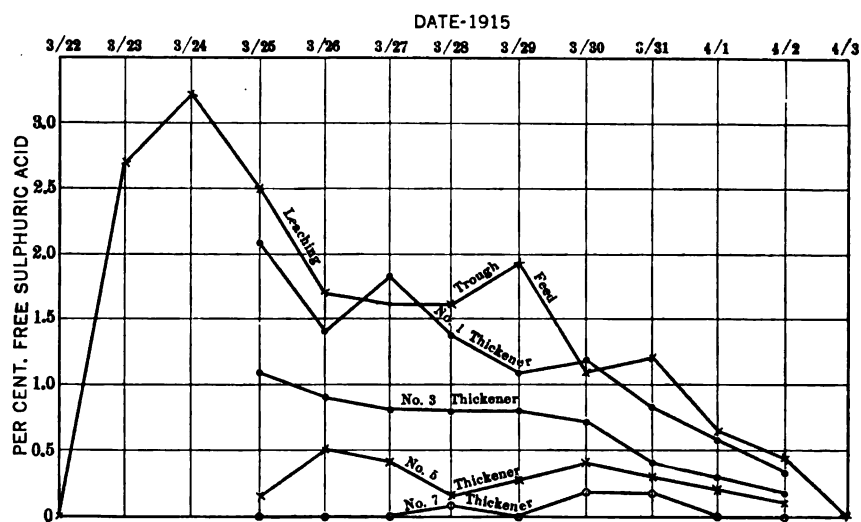


FIG. 14.—ACID IN LIQUORS.

the head of the sluice. The furnace arms alternated with light and heavy charges through the bottom drop hole and these were spread out over a reasonable time interval by placing an adjustable gate in the delivery spout, something on the plan of the classic hour glass. This worked fairly well, but it would have been better to have arranged some extra arms on the bottom hearth.

The copper content of the tailings at various points in their travel through the plant is shown in Fig. 13. The free sulphuric acid content of the liquor is shown in Fig. 14 and the effect of this strength of acid upon total extraction in Fig. 15.

The slimes as received assayed 0.84 per cent. of copper. Fig. 13 shows that the calcines averaged about 0.74 per cent. of copper, the grade falling as the furnace was gradually speeded up. This difference in assay was carefully investigated in the laboratory and it seems very improbable

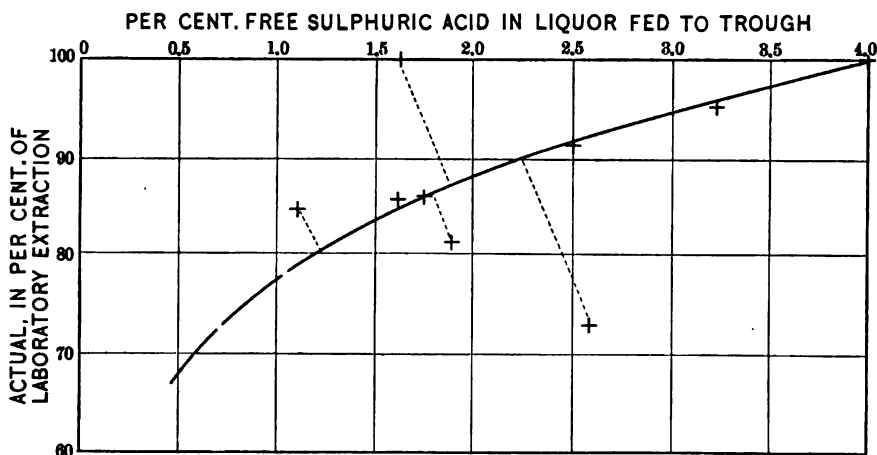


FIG. 15.—FREE ACID IN LIQUOR VS. EFFICIENCY OF EXTRACTION.

that it was due to the formation of insoluble copper compounds during the roast, as three acids were used in the assay and various methods yielded but the merest traces of copper from the assay residue. The most probable explanation lies in the formation of particles of anhydrous copper sulphate during the roast, which blew over with the flue dust. No measurements were made of the flue dust, as the furnace discharged into a common flue with other roasters treating concentrates.

The tails, as determined in the laboratory by heating samples of the calcines in 4 per cent. sulphuric acid, showed 0.2 per cent. of copper, gradually dropping to 0.1 per cent. as the furnace efficiency increased with the load. The actual final tails discharged from the last thickener ran about 0.1 per cent. higher than the laboratory-leached calcines and laboratory test showed this 0.1 per cent. to be present as soluble copper. This same

margin has been noted by Croasdale in his paper presented last year on his experiments at Ajo, and he there attributes it to the impossibility in practice of washing ore free from soluble salts. Fig. 15, however, seems to indicate that the difference in this case is largely due to the stronger acid liquor used in the laboratory tests.

Fig. 14 shows that the run was started with over 3 per cent. of free sulphuric acid at the leaching trough and that this strength was allowed to exhaust itself gradually upon the entering ore as the run proceeded, the difference between final actual tails and laboratory tails increasing as the acid dropped. This same ratio is the basis of Fig. 14. It seems probable, therefore, that the strength of acid used will have to be considered in its bearing upon efficiency of extraction.

The leaching trough discharge ran but a few hundredths of a per cent. higher in copper than the final tails after seven washings, showing that over 90 per cent. of the leaching was done in the few seconds taken to travel down the trough. This high efficiency is due to the violent agitation when the hot calcines meet the relatively cool liquor, and to the fact that each particle of solid meets a fresh entering drop of liquor at the head of the trough, obtaining the full efficiency of the acid strength. An actual over-all extraction in practice somewhere in the seventies seems assured from these experiments.

Washing

The separation of the dissolved values from the exhausted slimes was one of the main problems presented. Any method of percolation was out of the question owing to the extremely fine state of subdivision of the material. The acid liquors employed similarly ruled out many of the mechanical filters. It was finally decided to try out continuous counter-current decantation with Dorr thickeners of the acid-proof type of construction, some experience with which had been gained at the installation at the Butte-Duluth Mining Co.'s plant. Two 30 by 10 ft. and five 24 by 10 ft. tanks with suitable mechanisms and wood-stave pipes and fittings for 4-in. air lifts were therefore ordered and connected after erection in accordance with the flow sheet shown in Fig. 2.

The acid proofing of these mechanisms consisted of the substitution of wood for iron in the under-liquor parts. In turn, this called for some changes in design, two arms being used in place of four and the inclination of the swept floor being lessened. Both these changes increased the digging load and the arms tended to ride rather than dig, extreme cases overstraining the unit. In spite of these difficulties the thickeners proved a practical apparatus. I believe, however, that the original metal construction properly protected by lead and equipped with Duriron rakes would be an improvement. An idea of the wooden construction can be obtained from Fig. 16.

The main difficulty was with the air lifts and I am satisfied that these are not practical for the particular case. It must be remembered that it was desired to make an underflow of 50 per cent. solids and that in the first thickeners many small lumps of slime had to be handled. When an air lift was gradually throttled until the desired thickness was obtained it no longer acted as a pump but more like a cannon, and a small capacity was obtained, finally resulting in one of two undesirable climaxes. Either a few small lumps settled out in the horizontal pipe under the tank, due to the slow flow, thereby choking it, or the lift would run thin for a moment followed by a rush as the normal pumping action was restored, stirring up the settled slimes and requiring constant attention for the next half hour in order to get the desired thickness of underflow. Also, the scouring action on the wood elbow at the point where the air was introduced was excessive, pipes sometimes wearing through in 24 hr.

Simple spigot discharge into a launder with some mechanical means of elevating was not promising, again due to obstructions making the flow



FIG. 16.—THICKENER PADDLE.

of such a thick pulp very hard to regulate. Finally upon Mr. Dorr's suggestion two changes were made: (1) a diaphragm pump was tried, and (2) the remaining air lifts were arranged to return a part of the underflow to the same tank, thereby increasing the solids moved and keeping the rate of flow in the horizontal feed pipe above the critical speed at which lumps settled out.

The diaphragm pump was an entire success. In the first place, the delivery depends almost entirely upon the stroke and speed and is not influenced by a momentary change in thickness as in the case of the air lift. Then the suction has the entire pressure of the atmosphere at its command and this is exerted whenever an obstruction is encountered. In fact, an ore sack and a pair of gloves were pumped up, to the surprise of the operator. It is astonishing how many things find their way into a

system of this kind. A watch and fob and a brace and bit were among the articles added to our museum. Finally, the solids are moved steadily along and the scouring action is reduced to a degree which seems to make the use of wood pipe and fittings entirely practical. The pump was an ordinary 4-in. Gould contractor's pump with cast-iron body. The metal parts could readily be made of Duriron or other acid-resisting material.

During the run the temperature of the liquor in the first thickener gradually rose from 56° F. at the start until 100° F. was reached on the sixth day, at which point it was maintained by the sensible heat of the entering calcines. It was found possible to settle as high as 125 tons of solids in 24 hr. in one of the 30-ft. tanks. The overflow by actual test showed but 3 lb. of solids to the ton of liquor.

The liquors during the run accumulated sulphates corresponding to 10.0 lb. copper, 4.3 lb. iron, and 16.2 lb. alumina per ton of slimes leached. The actual acid consumed was 49.0 lb. The acid equivalent of the corresponding metallic oxides is 70.5 lb., showing the presence of considerable sulphates in the calcines. A laboratory test on the calcines, using 4 per cent. acid, showed a consumption of 68.9 lb. and a similar test on the product of a small-scale roast 69.1 lb. acid per ton. It appears, therefore, that less acid will be used in practice than in laboratory tests. Unfortunately, the run was not sufficiently prolonged to determine whether the proportion of impurities dissolved decreased as the liquors became more concentrated.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Commercial Production of Sound, Homogeneous Steel Ingots and Blooms

BY EMIL GATHMANN,* BALTIMORE, MD.

(San Francisco Meeting, September, 1915)

THROUGH wide experience at numerous mills in the United States I have found that there is a decided difference of opinion among the producers of steel as to what constitutes commercially sound steel. Some metallurgists insist upon having, for certain grades of steel, a so-called "open steel," *i.e.*, one which contains numerous blowholes of varying sizes, and consequently but little volume of true pipe, although a central intermittent shrinkage cavity often extends well down into the ingot. Such blowholes and pipe are expected to weld up during rolling or forging, because their surfaces are not oxidized. Even if this be admitted, included slag particles and a high degree of segregation are bound to be present in the product of such ingots. Blowholes are the result of an oxidized heat, and all heats, unless made or finished in the electric furnace, are oxidized more or less. Subsequent deoxidation, or rather degasification, either in the ladle or in the mold, or in both, is necessary to reduce or prevent blowholes. As is well known, a greater yield of billets or sheets is secured from the ingot by allowing the formation of blowholes, but this is undoubtedly obtained at the expense of the quality of the product. Decided segregation of carbon, phosphorus, and sulphur, as well as small included slag particles, undoubtedly exists throughout the greater portion of all so-called open steel, and cannot be removed by cropping.

The first requisite for sound, homogeneous steel ingots and blooms is therefore, in my opinion, so to treat the steel in the furnace, ladle, and mold that "piping" steel is produced. The line of demarcation between harmful and so-called "harmless" blowholes is practically impossible to define. Is it not, therefore, the better and safer practice to use means for the elimination of blowholes and correspondingly reduce the segregation and allow the formation of a well-defined shrinkage cavity or pipe at the upper end of the ingot? With the relatively cheap deoxidizers available at the present time, *e.g.*, ferro-silicon, aluminum and titanium,

* Manager, Gathmann Engineering Co.

there is no commercial reason why all steel should not be thoroughly degasified.

A degasified steel being provided, the method of freezing or solidifying the liquid steel into an ingot, and the subsequent working into blooms and thence into various products, is of great importance in reducing the crop or scrap portion of the steel, due to segregation and pipe. How to accomplish this reduction at a minimum expense and without upsetting the administrative mill practice of present plants is the problem. It has been my experience that the solidifying of an ingot made from steel which has been practically deoxidized or "killed" in the ladle, or in the mold,

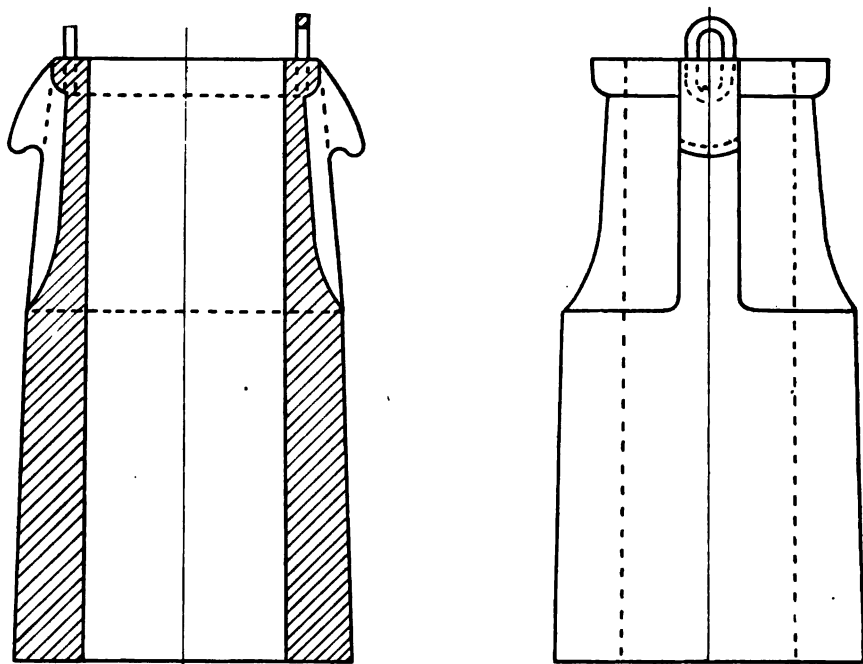


FIG. 1.—GATHMANN DESIGN OF INGOT MOLD.

depends upon the shape of the horizontal cross-section of the ingot at its various planes from top to bottom, and also upon the thickness and consequent heat-absorptive power of the various parts of the mold walls. An ingot with its larger horizontal cross-sectional area at the top is without question the best shape for obtaining the important "lag" in solidification of the steel, and whenever such large-end-up ingot can be conveniently used it is the best practice to do so. Nearly all of our large mills are, however, so equipped for handling and stripping the ingots that it is practically impossible to do this without extensive changes in equipment.

It is, therefore, necessary to employ means in ordinary big-end-down

ingots to greatly accelerate the freezing and solidifying of the lower and middle portion of the ingot and thus provide liquid metal to compensate for the decrease in volume caused by the contraction during solidification of the ingot. This I have accomplished by giving the lower 70 to 80 per cent. of the metallic mold in which the ingot is cast a much greater degree of heat-absorptive capacity than the upper 30 or 20 per cent. thereof. At first I attempted to accomplish this by ribbing the lower exterior surface of the mold in order to provide increased radiating area, or by water or air cooling the lower and middle portions of the mold. These earlier constructions were, however, found not to be practicable for commercial use. The desired results in differential cooling were then obtained by greatly thickening the lower and middle portions of the mold walls, consequently increasing their heat-absorptive capacity, and making the upper



FIG. 2.—TRAIN OF GATHMANN 20 BY 24 BY 80 IN. INGOT MOLDS.

mold wall thinner and less heat-absorptive. This construction will be best understood by referring to Fig. 1, showing a general design of mold, and Fig. 2, showing a train of molds of this type. Many hundreds of molds of this kind have been in daily use under usual furnace and mill conditions during the past year. Designs of molds have been made and established in actual practice for from 1- to 10-ton ingots.

In order to insure an ample supply of liquid steel to compensate for the shrinkage in the upper portion of the ingot in very dense, quick-setting steel, it was found advisable to use walls of a material of poor heat conductivity in the uppermost portion of the mold to supplement the differential effect obtained by the combination of the heavy and thin mold walls; in other words, to provide a sink-head.

The usual type of sink-head resting on top of the mold, or rigidly secured in the upper part of the mold cavity, was found not to fulfill all the conditions necessary in a commercial sense, primarily because of the high cost of such sink-heads when efficient. Sink-heads of this kind must be of sufficient thickness and strength to hold the liquid steel without cracking or breaking. Even a slight crack or break in the sink-head is sufficient to form a fin of steel and hang the ingot during its vertical shrinkage, thus causing surface cracks and defects in the body thereof.

Fig. 3 shows a method of freely suspending within the mold cavity a sink-head made of poor heat-conducting material. This type of sink-

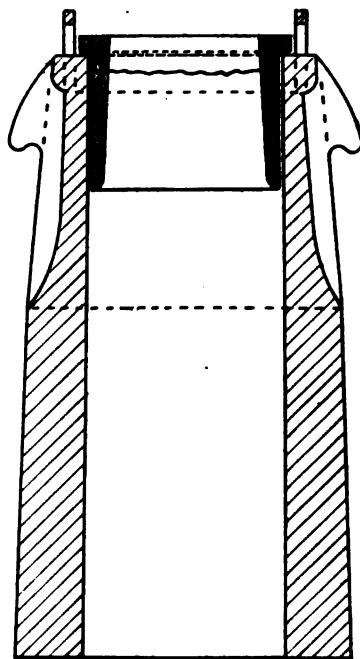


FIG. 3.—SINK-HEAD APPLIED TO GATHMANN INGOT MOLD.

head, in conjunction with the heavy-wall mold, has given most excellent results, and is in regular use by several of the largest high-quality steel works in this country, in conjunction with the mold having walls with a heavy body and thin top. Such a mold, if properly designed, will produce a sound homogeneous ingot with about a 20 percent. top crop, which is about 15 per cent. less than is usually necessary in ingots of like grade of steel made in the old-type mold. Figs. 4 and 5 show photographs of two 2-ton split ingots of 0.85 per cent. carbon, dead-killed steel, made in the same group, bottom cast, in Gathmann and in the old-type molds respectively. Split ingots of vanadium-nickel-chrome steel, one cast in a mold of the

standard type and the other cast in a Gathmann-type mold, are shown in Figs. 6 and 7.

With a sink-head in combination with the type of mold described the crop necessary to eliminate pipe and segregation is approximately 10 to 12 per cent. of the cold ingot.



FIG. 4.—SPLIT INGOT, MADE IN GATHMANN MOLD.

FIG. 5.—SPLIT INGOT, MADE IN REGULAR TYPE OF MOLD.

When the ingots are allowed to solidify in the mold the pipe will be formed to its maximum depth. I therefore advocate a fairly early stripping and rolling of the ingot, the actual time of stripping being governed by the cross-section of the ingot. No expensive change in admin-

istrative practice is required in the production of ingots by this method, which is in use by many steel plants, and it may be readily incorporated in the practice of any of the large steel works without upsetting in any manner the casting or mill practice.

The physically homogeneous condition of the ingot is obviously of

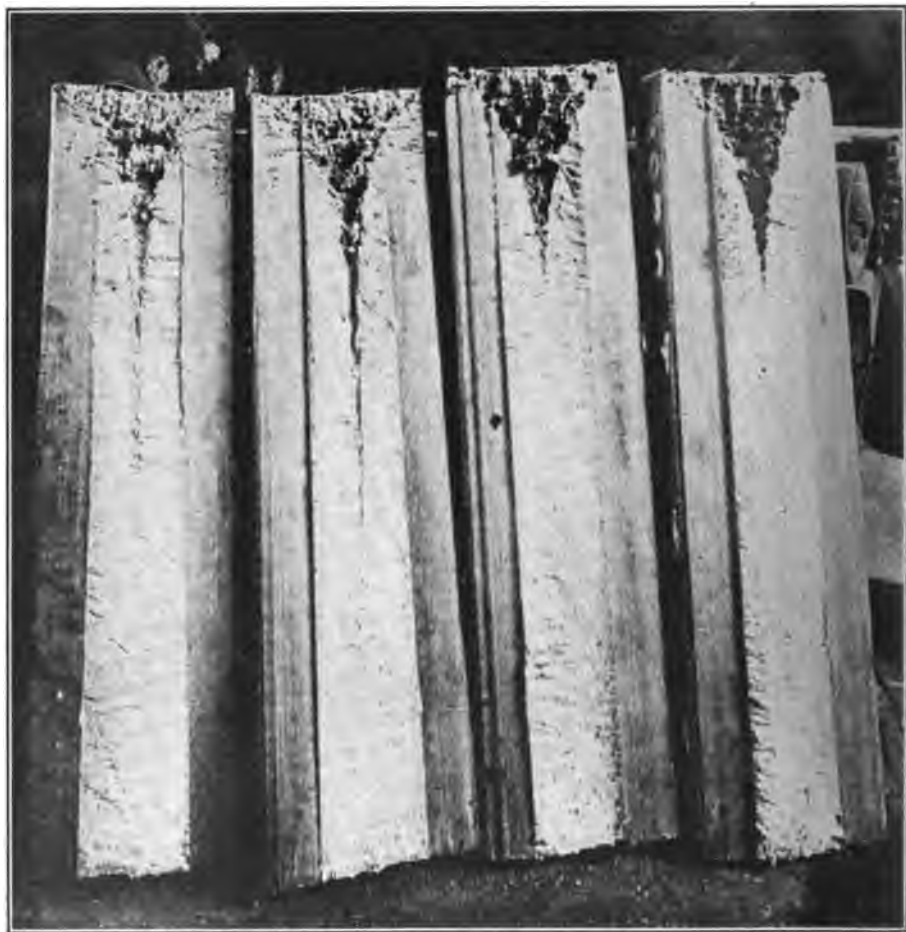


FIG. 6.—STANDARD INGOT OF VANADIUM-NICKEL-CHROME STEEL.

FIG. 7.—GATHMANN INGOT OF VANADIUM-NICKEL-CHROME STEEL.

primary importance in producing sound blooms and finished products. The method of blooming such ingots, is, however, also of importance. It has been my experience that in some heating and rolling practice there is danger at times of actually forming a false pipe, or rather fissure, in the ingot. This is especially the case where the ingot has been allowed to solidify and become cold in the mold. Many mills have found it good

practice to strip the mold from the ingot before its central portions have entirely solidified and place it immediately in the soaking pit or heating furnace in a vertical position, allowing the temperature to become equalized and the final feeding, due to shrinkage, to occur while in the soaking pit. The ingot is then rolled while the interior is hotter and more plastic than the exterior. *Great care* must, however, be exercised that the ingots be allowed to remain in the soaking pit until the steel is entirely set, because rolling with any green steel in the ingot will produce spongy centers. It is, therefore, necessary to determine the proper heating period before pulling specific sizes of ingots for blooming.

The best method of reducing ingots to blooms is without doubt to effect most of the reduction in butt-to-head passes, and I have seen a marked improvement in crop reduction obtained by this method of rolling,

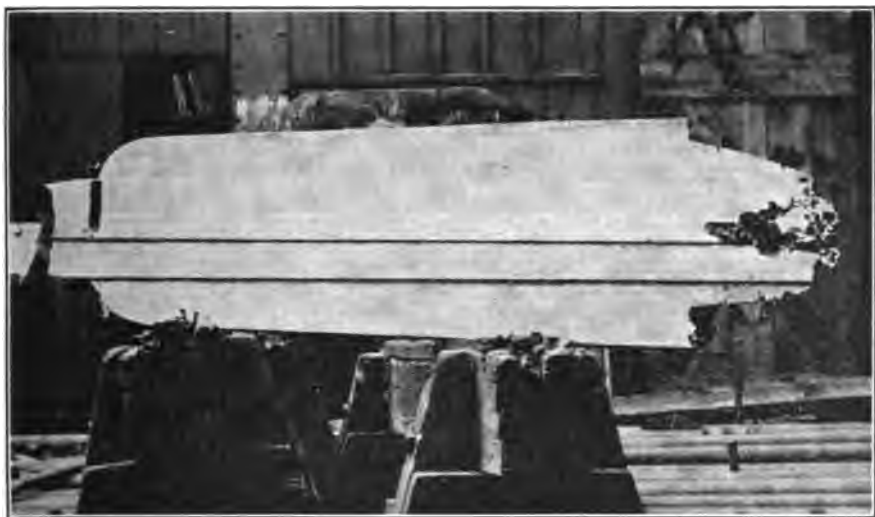


FIG. 8.—ONE-TON INGOT, MADE IN GATHMANN BIG-END-UP MOLD.

especially when the piped section was confined to the upper 20 per cent. of the ingot. Irrespective of any heating and mill practice which it may be desired to use at any specified mill, the physical condition of *cold ingots* split open for inspection is undoubtedly the true index to the value of any method for producing sound, homogeneous steel.

The theoretically ideal method of making sound, homogeneous steel is undoubtedly with the big-end-up mold, because the increasingly large area of the ingot toward the top compensates automatically for any irregularities in the temperature of the steel or in the teeming practice. The method I propose for utilizing the big-end-up mold in car practice was fully described in my previous paper.* Fig. 8 shows a split ingot

* *Trans.*, xlv, 461 to 472 (1913).

produced by my method under normal conditions with the big-end-up practice, the crop for piped section being approximately 5 per cent.

In the commercial production of sound ingots and blooms, tonnage and quality must, however, be jointly taken into consideration. I would, therefore, recommend in the usual mill practice where cars or bogies are used for carrying the molds, the heavy-walled body, light-walled top, big-end-down mold, as shown in Figs. 1 and 2 of this paper. I would also recommend the use of a suspended sink-head of non-conducting material, one type of which is shown in Fig. 3, to fully compensate for the contraction in the upper portion of the ingot. The molds and methods as described are applicable for the production of any grade of steel cast in ingots, and will invariably improve the quality and lessen the top crop. Thousands of tons of these molds have been made of both direct and cupola Bessemer iron, and have given excellent service as to number of heats per mold. Reports from mills using these methods and molds show that an important saving has been effected thereby in reduction of crop ends, and in making good steel better, in all cases where the molten steel has first been made sound by thorough deoxidation, which latter I consider the prime essential for the production of commercially sound, homogeneous steel.

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Lead Smelting at El Paso

BY H. F. EASTER,* B. A., EL PASO, TEX.

(San Francisco Meeting, September, 1915)

THE lead department of the El Paso Smelting Works at El Paso, Tex., is operated strictly on a custom basis. The ores treated are drawn from the surrounding territory, comprising New Mexico, Arizona, western Texas, and northern Mexico. Ores received from the last-named section originally amounted to a very large tonnage, but of recent years have dwindled into comparative insignificance so far as this plant is concerned. This has been due partly to the establishment of a lead-smelting plant at Chihuahua, which is much closer than El Paso to most of the important mines, and partly to the recent revolutionary troubles in Mexico. There are no mines of either lead or copper in the immediate vicinity of El Paso.

In general, it may be said that the lead-ore supply of recent years has been uncertain and precarious, with a general tendency toward an excess of silica and a shortage of iron. This latter requirement has been met in large part by the use in El Paso lead furnaces of considerable amounts of roasted leady copper matte from the Chihuahua smelter and of converter slag from the El Paso converters. These two materials, coupled with complex lead-copper ores from Bisbee, Ariz., have introduced into the lead-smelting operations a most unusual amount of copper, which has had an important bearing upon the smelting practice of the plant. This matter will be discussed later.

The ore, loaded in either box cars or gondolas, enters the storage yard on four tracks elevated about 8 ft. above the ground. As all unloading is done by hand, unloading platforms at the height of the car floor parallel the tracks on each side to afford a runway for wheelbarrows. At frequent intervals these lengthwise platforms are joined together by others running crosswise to facilitate the distribution of the ores evenly over the beds, which are located below and between the tracks. At present much of the bedding space formerly used for lead ores is given over to the copper department, but 20 bins of various sizes are retained for the use of the lead furnaces and the Huntington & Heberlein roasting plant. The largest of these holds about 1,000 tons, while the smallest holds not over

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175 tons, their combined storage capacity being about 10,000 tons. The bins have vertical wooden walls 8 ft. high and have no facilities for the removal of the ore by any mechanical means. Ore which for any reason cannot be accommodated in the bins is unloaded in stock piles in the outer yards and eventually loaded into railroad cars again with a Browning locomotive crane, which has been a part of the plant equipment for about two years past.

All unloading is done by contract, the rates per ton being based upon the character of the car from which the ore is taken, and the distance to which it must be wheeled. When, as is generally the case, the initial sampling of the ore is done by hand, the contract rate includes the cutting of the sample and the wheeling of it to the sample storage platform. An extra allowance is made for spreading the ore in the bin itself, which in most cases is carefully done by hand because of the widely different character of the various small lots which are bedded together. This last-mentioned condition is typical of the El Paso plant and the large number of small shipments has always been a serious obstacle in the way of installing a modern ore-bedding system.

In most cases it is customary to assign to each car to be unloaded a gang of four men, who cut with their shovels a one-tenth sample from the fines and set aside all lumps over about 6 in. for sampling in the automatic sampler, which occupies a space on the platform adjacent to the ore bins. This automatic plant is equipped with Vezin samplers, which are arranged to cut a one-tenth sample that may be combined with the shovel sample of the fines. The full equipment includes one 75-hp. motor; one No. 6 Gates crusher, capacity about 45 tons per hour; one 14-in. bucket elevator; one Vezin sampler, 52 in. in diameter over the wings, making a one-fifth cut; one set of 14 by 36 in. rolls, adjustable to $1\frac{1}{2}$ in.; one 10-in. bucket elevator; two smaller Vezin samplers, of which the combined sample cut from the stream is one-half.

The sample of the lumps from the automatic is added to the hand-cut sample of the fines and this combined sample is wheeled to one of the two smaller automatic samplers facing the unloading platforms and just inside the sample room proper. These two mills are practically identical. Their equipment includes one 9 by 12 in. Blake crusher; one 8-in. bucket elevator; one Vezin sampler cutting one-fifth; one set of 12 by 36 in. rolls; one 6-in. bucket elevator; and two Vezin samplers so placed that each one makes a one-fourth cut at the same point. These mills deliver two identical samples, crushed to $\frac{1}{2}$ in., which are generally finished up and assayed separately. These samples weigh from 400 to 500 lb., in the case of a one-car lot. After going through the finishing rolls, of which there are two sets, 14 by 24 in., the samples are wheeled to a separate compartment which has an iron-plated floor. There the samples are circled, coned, spread, and quartered by hand until reduced to a proper size to

send to the bucking room. A portion of the reject from the plates is set aside and held pending the final settlement with the shipper, in case any doubt may be cast upon the accuracy of the work done in the bucking room.

The foregoing is the usual method of sampling lead ores, although exceptions occur in the case of shipments of matte or coarse lump ore, and in the case of high-grade ores and concentrates. In the case of the former, the shovel cut is dispensed with and the whole carload is put through the large automatic sampler, while in the case of concentrates the procedure is reversed and the entire sample is cut by shovel one-tenth or one-fifth according to the grade, the sample being reduced entirely by coning, circling, and quartering. Very high-grade ores are usually cut one-fifth.

After sampling, all sulphide ores and leady copper mattes destined for treatment in the Huntington & Heberlein roasting plant are wheeled to the sulphide mill, which occupies a space on the unloading platform between the sample room and the roasters. This mill contains the following: One 12 by 18 in. Blake crusher; one set of 16 by 30 in. rolls, crushing to $\frac{1}{2}$ in.; one 12-in. bucket elevator; one $3\frac{1}{2}$ by 8 ft. trommel having openings $\frac{3}{4}$ by $\frac{3}{16}$ in.; one set of 6 by 42 in. high-speed rolls, to take the oversize from the trommel; one 8-in. bucket elevator, to return same to trommel again. This mill has a capacity of about 11 tons per hour.

The leady copper matte is generally bedded by itself, while the crushed ore is often bedded with sulphide concentrates, of which a varying tonnage is received at the plant. Because of the uncertain nature of the ore supply, it is difficult to say just what might be considered a typical mixture for the roasters. In general, the best results are obtained from a mixture of ore and leady copper matte, as straight ore is difficult to handle alone, and it has been found necessary to use at least 20 per cent. of siliceous sulphides to produce satisfactory results when roasting leady copper matte.

The ore and leady copper matte are trammed in 1-ton cars from the bins to the roaster plant, where the cars are elevated by a small hydraulic elevator to the level of the charge floor of the Godfrey roaster. The material is dumped into conical hoppers that feed mechanically into three Godfrey roasting furnaces, which are equipped with revolving hearths and stationary rabblers which plow the material gradually from the feed at the center to the discharge hole at the outer edge. All three furnaces were originally fired with coal which has since been replaced in the case of two of them by oil. The hot coals from the fire-box of the third furnace are used to start the Huntington & Heberlein pots. These Godfrey furnaces are 26 ft. in diameter and have a capacity of about 30 tons each per 24 hr. The oil-fired furnaces use about 0.22 bbl. per ton roasted, while the coal-fired furnace uses about 0.11 ton of coal, all of which, how-

ever, is not chargeable against direct roasting, as much half-burned material is drawn off for the pots. The initial charge probably averages 21 per cent. sulphur and this is brought down to from 10 to 12 per cent. in the Godfrey roasters. Care is taken not to make too strong a roast at this point, or the mixture will not work well in the pots.

The roasted material is slightly moistened by a spray as it leaves the discharge hole of the roaster and falls into a small hopper. From the hopper it is drawn into cars, trammed to the elevator, and raised to the charging floor of the Huntington & Heberlein pots. There are eight of these, each of which handles two charges of about 6 to 7 tons in 24 hr. At times it has been found possible to add to the pre-roasted material in the pots sufficient raw sulphides to bring the total capacity of the plant up to 100 tons per 24 hr. It is customary to work the pots in sets of four, so that when one set is charging the other is nearing the end of the blow. When the pots are filled, which takes about 2 hr. for the four pots, the full pressure of the blast is thrown on while at the same time the blast is cut very low on the four other pots which are finishing. Each charge is blown for about 11 hr., then dumped, and 1 hr. more is consumed in preparing for the next charge.

Air is furnished to the pots at a maximum pressure of about 10 oz. by a No. 7½ Baker blower, belt connected to a 35-hp. motor. The pots are suspended at a height of 10 ft. above the ground level and are dumped by a worm gear operated by hand. The sintered material is broken more or less by the fall, after which the breaking is completed by hand. The lumps are reduced to about 8 in. size, loaded into wheelbarrows, elevated to the level of the ore-unloading platforms by means of a small hydraulic elevator, and dumped into one of the regular ore bins convenient to the blast-furnace charge floor. The fines, which amount to from 5 to 10 per cent., are gathered up and returned to the pots on the next charge. The sulphur in the final product going to the bins averages about 5 per cent. The work of breaking and wheeling to the bins is done by contract.

The ore beds are grouped in the rear of the blast-furnace building and are connected with the weighing floor by a system of hallways, through which the ore is brought in wheelbarrows to a hopper standing over the larry-car tunnel. These tunnels, of which there are two, run at right angles to the line of blast furnaces and lead on to hydraulic elevators that raise the cars to the level of the feed floor, a lift of 41 ft. Because of the angle of approach it is necessary for the cars to run around a sharp curve after reaching the feed floor in order that they may pass lengthwise over all the furnaces in turn. As the elevators are approximately in the center of the building this operation involves a switch to reach the furnaces in the middle of the line. The larry cars are of the dump-bottom type, 5 by 10 ft., with a capacity of about 200 cu. ft., and are cap-

able of handling a 5-ton charge, although the charge actually in use at the present time averages about 4 tons dry weight including the weight of the coke. The cars are operated by 4-hp. motors connected with a third rail, and are hand dumped by a system of levers and counterweights.

The charge, which usually contains an average of about 8 per cent. moisture and is figured for about 7,000 lb. dry weight, is handled by wheelbarrow in units of from 400 to 500 lb., the latter being considered about the maximum load for the average wheeler. The various ingredients of the charge, after being carefully weighed upon ordinary multiple-beam platform scales, are distributed as evenly as possible in the charge hopper. This hopper receives the ore, both raw and roasted, and the limestone, which latter arrives at the plant in 50-ton dump-bottom railroad cars and is dumped through a trestle near the scales. After receiving the main portion of the charge from the hopper, the car moves under an open hatch about 15 ft. distant, through which are dumped the coke, scrap iron, dry dross, and miscellaneous furnace products that are being added to the charge. The coke is usually trammed direct from the cars in which it is received at the plant.

On the charge floor each furnace is provided with two parallel openings 1 ft. 4 in. by 11 ft., closed by counterweighted steel doors that may be easily opened by means of levers at the end. The charge car stops directly over the furnace and dumps its load, which is distributed by the spreaders below.

There were originally eight lead blast furnaces, but two of them were dismantled in 1913 to make room for an additional copper furnace.

The furnaces have the usual brick base 7 ft. 2 in. by 15 ft. 5 in. inside the crucible plates, which are 3 ft. 4½ in. high and stand upon a flanged steel pan. The crucible proper is 24 in. wide at the bottom, widening to 3 ft. 6 in. at the top. It is 12 ft. 9 in. long and 2 ft. 6 in. deep. The lead well is 12 in. square, and is built up 9 in. above the top of the crucible plates. It is on the right-hand side, looking toward the furnace, and its center comes 4 ft. 3 in. from the end crucible plate.

Resting upon the brickwork surrounding the crucible is the tier of lower jackets, which are 46 in. in vertical height and have a bosh of 9 in. at both ends and sides. The lower side jackets are of steel, all others, both upper and lower, being cast iron. Between the lower and upper tier is a 3-in. layer of firebrick. The upper jackets are 24 in. in height without bosh. Each of the lower side jackets, four on a side, has two 3½-in. tuyères placed 13 in. from the center of the hole to the bottom of the jacket. The hearth area of the furnace at the tuyères is 46 by 162 in., or 47¼ sq. ft.

Supported by brick arches which rest upon 8-in. iron columns at each corner of the furnace is the brick shaft, 10 ft. 8 in. by 20 ft. 4 in., outside

measurement, and extending upward a height of 23 ft. to the feed floor. Inside the shaft the furnace widens from 5 ft. 6 in. at the top of the jackets to 7 ft. 1 in. at a point 6 ft. below the charge floor, which is about the level at which the ore column is usually maintained.

The furnaces average about 180 tons each of total charge per furnace day, although with a coarse charge they have frequently done better than 200 tons for considerable periods. During recent years not more than three furnaces have been in operation at any one time.

At the rear of the furnace just under the floor plates, a rectangular steel downtake, 3 by 6 ft., conducts the gases to the brick flue chamber, a vertical distance of 11 ft. 4 in. and a horizontal distance of 12 ft. 4 in. The flue system, which varies somewhat in the different sections, has an area of 151 sq. ft. at its smallest cross-section and 170 sq. ft. at the largest. Its total length is about 800 ft., of which the first 375 ft. nearest the furnaces is provided with rectangular iron hoppers overhead that discharge into tram cars on a track within the walls of the flue chamber. The rest of the flue is provided only with iron doors at intervals along the sides at the ground level, through which the flue dust must be raked out. By far the greater amount of dust is deposited in the hoppers, from which it is removed daily, while the rest of the flue is ordinarily cleaned only once in three months.

The production of flue dust is not large, averaging from $1\frac{1}{2}$ to 2 per cent. of the charge smelted. The following average analysis for a recent month gives a fair idea of the ordinary composition: SiO_2 , 16.8; Fe, 17.6; Mn, 0.6; CaO , 8.1; Zn, 4.0; S, 9.0; Pb, 18.2; Cu, 1.4; Au, 0.30; Ag, 23.5 per cent. The flue dust is briquetted with 10 per cent. burned lime in a Chisholm, Boyd & White press, which has a capacity of about 4,000 briquets or 6.5 tons per hour. The briquets are stacked on wooden pallets, 300 to the pallet, and are taken to the drying shed on Scott briquet cars, which are so arranged that the carrying platform may be raised and lowered by a lever so as to permit the car to be easily withdrawn from under the pallet, leaving the latter resting on the stands. The shed has a storage capacity for 425 tons, which is sufficiently large to permit the briquets to be left to dry for from three to six weeks. After this length of time they are exceedingly hard and reach the furnace in very good shape.

For a long time past the settling and separation of the leady copper matte from the slag has been a most vexing question for the El Paso plant. This was originally accomplished by the use of a Rhodes "separator," or small reverberatory settling furnace, which in the days of large tonnage was generally satisfactory. However, with the present comparatively small tonnage of lead ores smelted it has proved increasingly difficult and expensive to successfully operate these furnaces and they are now permanently abandoned. In spite of this, the "separators"

were for so long a distinctive feature of the plant that it may not be out of place to give a brief description of them and of the details of their operation.

There were two of these furnaces placed end to end in a line parallel with the line of blast furnaces and about 100 ft. distant from it. It was customary to operate only one separator at a time, the other being held in reserve. The slag and leady copper matte were tapped from the lead furnaces directly into small pots, which were then pushed by hand along iron-plated runways to the dump hole of the separator. Also for many years all the slag from the settlers at the copper blast furnaces was put into the separator, partly because it helped to keep the furnace in better condition and partly because there was no other easy way of disposing of it.

These settling furnaces had an effective inside area 14 by 25 ft., with rounded corners at the end opposite the firebox. The whole structure rested upon a steel pan supported by 6-in. I-beams. Above the pan came 9 in. of tamping, in which was imbedded a continuous coil of 1½-in. pipe covering the entire surface. Above the tamping was 9 in. of firebrick, laid in the form of an inverted arch with a dip of about 2 in. in the middle. The walls were 1 ft. 8 in. thick and contained a tier of cast-iron coil jackets, with a lining of 4½ in. of firebrick inside the jackets. The roof was formed of a 9-in. firebrick arch and sloped downward from the bridge wall toward the uptake to the stack at the opposite end. The dump hole was at the corner close to the uptake; the slag tap on the same side as the dump hole, but at the opposite end of the furnace near the bridge wall; and the matte tap just about opposite the slag tap on the other side of the furnace. A lead tap was also provided close to the matte tap on the same side. The ordinary depth of the bath of molten slag and matte was about 36 in. The center of the slag tap hole was 28 in. above the brick bottom, the matte hole 15 in., while the lead tap was level with the bottom at its deepest part. Both slag and matte taps were water jacketed. The slag was tapped direct into oval-bowl dump cars of about 3 tons capacity and hauled to the dump by a small steam locomotive. The leady copper matte was tapped into a rectangular cast-iron box 3 ft. 8 in. by 6 ft. 2 in., which served as a trap to catch any metallic lead that might come with the matte. From the box two cast-iron spouts conducted the matte to the edge of the granulating pit, which was a cement-lined reservoir constantly filled with water and from which the granulated matte was pulled with hoes. The two streams of matte falling from the spouts were cut by a double spray of water at about 100 lb. pressure, two nozzles being provided for each stream as a safety measure in case one might suddenly become clogged up. The granulated matte was wheeled to railroad cars and transferred to the Huntington & Heberlein roaster

plant, where it seemed to give quite as good results as matte that had been crushed in the sulphide mill.

The furnaces were originally coal fired, but during November, 1911, they were changed over to oil. At first the burners were introduced through the firebox and played over the bridge wall, being inclined slightly downward and exercising a reverberatory action. Afterward this arrangement was changed and the burners set to play directly downward through the roof, one at each corner of the furnace. This was a decided improvement and kept the furnace in much better condition with less oil. The amount of oil used varied considerably, but 20 bbl. per day was about the average.

The advantages of the separator were the almost perfect settling of the leady copper matte under normal conditions and its cheap preparation for roasting by means of granulation. The disadvantages were much more numerous. To begin with, the transportation of the material from the blast furnaces in hand pots was expensive. The practice of mixing lead and copper slags together in the presence of leady copper matte was eventually found to be a serious mistake and steps were taken to dispose of the copper slag elsewhere. It was conclusively proved that the copper slag saturated itself with lead and silver and thus carried to the dump valuable metals that would otherwise have been saved. The effect of cutting out the copper slag, the volume of which at times was considerable, was to leave the separator an easier prey to the troubles brought on by speiss crusts and zinc mush, and to increase the amount of fuel necessary to keep the bath in good condition. It was found that high zinc in the charge would fill up the separator with "mush" quite as quickly as it would a smaller settler and that a few days of exceptionally strong reduction on the blast furnaces would often produce layers of speiss that would set and form pockets and false bottoms that were equally fatal to the settling action of the furnace. Another peculiarity was that the leady copper matte invariably came out with a higher lead assay than it had when it went in. This undesirable condition was thought to be due to a reaction between comparatively large surfaces of molten matte and metallic lead, of which there was always more or less at the bottom of the furnace.

It was never found practicable to operate the separator with less than four men on a shift, and when the lead plant got down to a two-furnace basis the amount of fuel burned was much greater than it would have been had a larger volume of molten material been handled through the settling furnace. The length of the campaigns of the separators between shutdowns for extensive repairs became shorter and shorter until in July, 1914, a newly repaired furnace became a hopeless mass of crusts and pockets within a week of the time it was put in use. The other furnace was in similar shape and the plant was left without any settling

device except small rectangular cast-iron boxes 3 ft. 4 in. by 6 ft. 4 in. and 2 ft. deep, which were entirely inadequate even when run in tandem. At this time every one connected with the plant became convinced that the separators had outlived their usefulness and that a change was imperative, but the whole blast-furnace system was linked to these furnaces in such a way that a change to any standard plan could be made only at an excessive cost and an unjustifiable interference with the operations of the plant in general. One serious difficulty was the lack of sufficient elevation in front of the blast furnaces, the space between them and the separators being comparatively level and only 4 ft. below the furnace tap holes. This was practically all made ground, having been built up of loose boulder slag, which made excavation difficult and dangerous on account of its tendency to ravel and cave in. Another trouble was the lack of room for any arrangement of tracks such as are generally to be found in front of a line of modern blast furnaces.

It was decided, therefore, to proceed along somewhat original lines, and on July 25, 1914, there was put into operation in front of one furnace a round, oil-fired settler, 8 ft. in diameter. This was intended to serve merely as a second settler to remove the last traces of matte from the slag before going to the dump. For the main separation one of the old rectangular boxes, already mentioned, was relied on, being placed between the round settler and the furnace. Results obtained from this settler were not particularly satisfactory as numerous particles of matte persisted in floating across the surface of the bath direct from intake to outlet. Also, the settler shortly began to fill up with a zinc mush that interfered seriously with proper settling. While the slag that was being run at the time contained only 4 per cent. of ZnO , there seemed to be present in the Huntington & Heberlein roast enough unaltered zinc sulphide to give trouble. The furnace to which this settler was attached was blown out soon afterward and was replaced by another furnace, in front of which was a similar arrangement of box and round tank, except that in this case the latter was 10 ft. in diameter. The tank was set off center far enough to give the furnace tapper a chance to work, which made necessary a rather long curved spout from the box to the tank. The greatest trouble encountered was the lack of elevation, as the round settler was erected on the original level of the ground in front of the furnace and it was necessary to have both slag and matte taps high enough for the hand pots to get under them. It was thus necessary for the slag to flow a total distance of about 23 ft. with a drop of only 11 in. between the tap hole and the outlet from the settler. This of course caused trouble every time the slag happened to get thick and pasty. The first settler, or box, was easily removable, had a firebrick lining (magnesite was afterward used with great success), and measured 66 in. long, 28 in. wide, and 21 in. deep inside the lining. The round settler had a 9-in. lining of

magnesite brick, both sides and bottom, and a 9-in. arched roof of fire-brick, upon which rested at the edge of the tank, just over the slag outlet, a short brick stack with an iron pipe connection above. The oil burner was placed at the exact center of the roof and was directed straight down upon the surface of the bath. A small combustion chamber was later placed on the roof and proved very successful in getting more efficient results from the oil.

It was thought that a baffle extending across the tank close to the inlet and partly immersed in the bath might compel the floating particles of matte to go down and join the main body of matte instead of going straight across and out with the slag. Accordingly, such a baffle, constructed of brick, was placed about 12 in. from the inside of the lining. This feature of the experiment was a complete failure and the floating specks of matte continued to pass out on the surface of the slag as before. These small particles, however, proved upon investigation not to involve so serious a loss as had been feared. Several hand pots of slag were set aside to cool, each of which contained a plainly visible number of the specks. After cooling, the balls were broken up and put through the small automatic sampler. The average results were: Silver, 0.65; metallic lead, 1.00; copper, 0.15 per cent.

After starting off favorably, this new settler soon began to fill up with zinc mush. Some of this material had a specific gravity lower than that of the slag, while some of it appeared to be slightly heavier, the tendency as a whole being to form a layer between the slag and the matte. Various samples of this material were analyzed, a typical one being as follows: Insoluble, 6.6; iron, 19; lime, 3.4; zinc, 24.3; sulphur, 21; lead, 10; copper, 4.6 per cent. When this crust was broken up with a bar, it floated for a while at or near the surface in lumps resembling dry dross. This was not a new condition, having frequently been observed in the Rhodes separators. The only way to overcome it seemed to be to let the settler fill up with matte, thus forcing the mush to the top, where it was skimmed off with rabbles through a hole cut for the purpose. This of course involved a considerable loss of matte and a large proportion of the slag made at this time had to be saved for resmelting. This condition of affairs was finally remedied by changing the charge so as to use less of the Huntington & Heberlein roast. After this time similar troubles were promptly met in their early stages by changes in the furnace and roaster charges and never again caused serious trouble while this style of settler was being used.

As already mentioned, the original idea with regard to this type of settler was to use it simply as a safeguard against letting any matte go to the dump, as much matte as possible being tapped directly from the box. It was found that this amounted to about 88 per cent. of the total matte produced. The shells from the hand pots in which the slag was

removed were frequently and carefully sampled, and they assayed so little higher than the regular slag samples that they were thrown away. The ordinary method of taking the regular daily samples was to take a small ladleful from the stream as it flowed from the round settler, once every hour, and to pour it into water for granulation. At the end of 24 hr. the whole sample was worked down for a pulp for the assay office.

A second furnace was blown in Sept. 29, 1914, with a settling arrangement exactly similar to the one then in use. The two furnaces were operated throughout the month of October in the manner already described. The average analysis of the slag for the month was: SiO_2 , 33.7; FeO , 27.9; MnO , 1.4; CaO , 22.9; ZnO , 4.6; Al_2O_3 , 5.2; Ag, 0.6; metallic lead, 1.50; Cu, 0.15 per cent. The leady copper matte produced averaged Fe, 42.1; Pb, 12.1; Cu, 19.8; Zn, 4.6; S, 21.4 per cent.; Au, 0.04 oz.; Ag, 87.8 oz. The average grade of the bullion produced was: Au, 3.24; Ag, 247.6. The matte produced averaged 8 per cent. of the total charge. Lead on the charge averaged 15.7 per cent., and sulphur 2.2 per cent. Coke used averaged 14.2 per cent. These figures, while applying to only a single month, may be taken as fairly representative of the lead-smelting practice of the plant.

The high copper in the leady copper matte, as shown in these figures for the month of October, is characteristic of the usual conditions at this plant. It had long been customary to set aside matte of about this grade until a sufficient amount had been accumulated to make worth while a concentration run. The matte produced during these campaigns usually averaged about 10 per cent. Pb and 35 to 40 per cent. Cu. It was drawn off in hand pots, cold dumped, broken by hand, and resmelted through the copper blast furnaces. All this involved considerable expense and it became evident that much of this could be saved if some means were provided for getting this molten leady copper matte direct to the converters. The round oil-fired settlers offered a convenient reservoir in which to collect the matte in fairly large quantities, and a pit 10 ft. wide and 8 ft. deep was dug immediately in front of the settlers in order to allow the approach of the 5-ton matte ladles serving the converters. About this same time the rolling stock equipment of the plant received a much needed addition in the shape of several Treadwell slag cars of 160 cu. ft. capacity, holding about 14 tons of slag each. These cars superseded the older ones of small capacity and made it possible to handle all the slag and matte away from the furnaces over a single straight track laid in the pit in front of the furnaces, entirely doing away with the necessity for hand pots for any purpose. This, however, involved a radical change in the use of the settling tanks, inasmuch as all the leady copper matte was run over into them and the rectangular box was used only as a collector for lead. Owing to the high percentage of copper on the charge and the large amount of dross produced, the lead wells of the furnaces are very

hard to keep open, and most of the time all the lead passes out of the tap hole and must be separated outside the furnace.

The direct conversion of the leady copper matte proved to be such an economy that since the introduction of this system, matte has been converted that is much lower in copper than formerly. At the time of writing it is customary to figure the value of the lead contained in the leady copper matte, and therefore lost in converting, as against the cost of roasting and resmelting in the lead furnace, taking into consideration the fact that each unit of copper present means the loss of a certain amount of lead even on a concentration run. Each day's production of leady copper matte is thus disposed of according to its assay, either direct to the converters or poured on the ground in beds, to be picked up, crushed, and roasted. As the matte has recently been rather low in lead, most of it has gone direct to the converters even though it may not run higher than 20 per cent. copper.

Under this system the whole duty of settling out the matte is put up to the large settler, and the rectangular box is used only to separate out the lead. Considerable anxiety has been felt as to whether it is safe to rely upon a single settler for the separation of the matte. In an endeavor to throw some light upon this question, the shells from the Treadwell pots have frequently been brought back from the dump and sampled with great care. The most complete of these tests covered a period of five full days early in January, 1915, all the shells produced being saved and run through the automatic sampler. The resulting composite sample showed: Au, trace; Ag, 0.64; metallic lead, 1.46; Cu, 0.30 per cent. The average of the daily 24-hr. samples for the same period showed: Au, trace; Ag, 0.30; metallic lead, 1.24; Cu, 0.24 per cent. These figures, which are about in line with all the others obtained under similar conditions, would indicate that it is safe to put these shells over the dump.

The comparative success attending the use of these round oil-fired settlers encouraged the installation of a somewhat similar but larger settler intended to serve two furnaces instead of one. The apparent advantages of this were the greater volume of leady copper matte that could be accumulated at one time, the certainty that one such settler would require less fuel oil than two of the earlier type, and some saving in labor.

This new settler is placed about 6 ft. forward of the line of the front crucible plates of the two furnaces and equidistant from them. It is 10 ft. inside by 20 ft. long, extending close enough to the edge of the train pit to make it easy to tap matte and slag into the cars below without the use of excessively long spouts. The whole structure is supported upon rails running crosswise and providing air spaces beneath the bottom pan. This feature has proved to be of considerable importance inasmuch as there is a constant seepage of lead through the bottom and the open air spaces arrest its progress downward into the foundation and facilitate its

removal. The ends of the settler are slightly curved in order to give a stronger structure. The sides are braced with I-beams set upright and tied across the top with steel rods. The sides and ends have a lining of 9 in. of magnesite brick. A matte tap is provided on each side close to the front, and there is a single lead tap in the middle at the rear end. This lead tap is used every day, but very little lead is obtained, as experience with the Rhodes separators indicated the inadvisability of maintaining a bath of lead in contact with the matte, and every effort is made to keep lead from getting into the large settler in the first place. The roof of the settler is sectional, being composed of a number of rail-bound firebrick arches that can easily be lifted off if necessary. The stack is situated at the rear end immediately between the slag inlets from the two furnaces and over the lead tap. Two oil burners were installed, one playing downward through the roof at a point about 6 ft. back from the front, and one playing nearly horizontal through the front wall.

Between each furnace and the large settler is a removable rectangular box, 2 ft. 6 in. wide by 9 ft. 2 in. long, lined with $4\frac{1}{2}$ in. of magnesite brick all around. The furnace is connected with the box and the box with the settler by short water-coil spouts. Each box is set at an angle of about 45° to the furnace, giving the tapper a clear field for his work.

As it is no longer necessary to provide for the use of hand pots under the slag and matte spouts, it has been possible to arrange the elevations of the various parts of the system to much better advantage than in the case of the round settlers. There is a difference of 20 in. between the tap hole of the furnace and the bottom of the slag outlet from the main settler, which greatly facilitates the handling of the spouts. There is a difference of 10 in. between the bottom of the inlet and the bottom of the slag outlet, which allows plenty of time for switching and changing slag cars after the outlet hole has been closed up. The normal depth of the bath in the settler is 25 in. with the slag outlet open, and it is customary to maintain about 10 to 12 in. of matte at all times, as this seems to assist materially in keeping the settler in good condition.

Although this settler was designed for operation with two furnaces, it was decided to try it out with only one, in order to see if it might be kept open under such circumstances. At the time of writing it has been in successful operation with one furnace for exactly one month, although the amount of fuel oil necessary to keep it in good condition is somewhat greater than was the case with the former round settlers. The larger settler requires about 7 bbl. of oil per day, whereas the others averaged about 4.5 to 5 bbl. However, with two furnaces running it is probable that less oil would be required. No serious troubles have been encountered up to the present time, and careful sampling both at the furnace and on the dump indicates that the settler is standing up to its work quite well.

At such times as the lead wells are in working order, the lead is drawn

off into cast-iron pots at the side of the furnace, drossed by hand, and the dross returned directly to the furnace. When, as is more frequently the case at present, the lead wells are not working and all the lead is taken from the box, no attempt is made to dross it at the furnace. Instead, the undrossed bars are trammed to a small sweater furnace, where they are remelted upon a sloping hearth, the clean lead drained off, and the dry dross raked out and returned to the furnace charge. This equipment is inconveniently situated and antiquated in the extreme, and it is hoped will be speedily abandoned.

Whether drossed at the blast furnace or at the remelting furnace, the bullion is sampled in the same manner, as follows: The bullion is tapped into a cast-iron cooling pot. The dross is skimmed off carefully and thoroughly. Before the final skimming the contents of the kettle are well stirred with the skimmer and the dross thereby raised is taken off completely. After skimming, the contents of the cooler are stirred again and, while the lead is still in motion, "gum-drop" samples are dipped. A few more gum drops are taken than it is expected to make bars of lead, to insure having a gum drop for each bar of lead. The bullion is ladled into molds, and after the pot is empty all extra gum drops over the number of bars produced are thrown out. These gum drops are held and delivered to the assay office whenever the bullion corresponding to them is removed.

A shipping lot consists of 780 bars (about 42 tons), and, when complete, the 780 gum drops corresponding are placed in a graphite crucible and melted at as low a temperature as will give a homogeneous mass from which to dip the final gum-drop samples, upon which the actual assay of the lot is made. Great care is taken not to get the sample hot enough to oxidize the lead during the process.

The gum drops, which weigh about 40 g. each, are assayed without clipping and the gold and silver calculated to an assay-ton basis.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Zinc-Dust Precipitation Tests

BY NATHANIEL HERZ,* E. M., LEAD, S. D.

(San Francisco Meeting, September, 1915)

THE use of zinc dust for precipitating the precious metals from cyanide solutions is well established now in many places, and has many advantages over the shavings method of precipitation. Although much work has been done on methods of testing zinc dust, as yet no absolutely positive test has been devised to determine the value of the material. Certain facts are known, but often it happens that results on a working scale are not as expected from the laboratory tests. The purpose of this paper is to record some observations made while testing a large variety of zinc-dust samples in the laboratory.

W. J. Sharwood¹ has described standard analytical methods, which have been followed, with the changes noted. The regular determinations include zinc oxide, by solution in ammonia and ammonium chloride, lead, and precipitating efficiency. In some cases cadmium was determined by a method similar to that described by J. E. Clennell.² I will discuss the precipitating-efficiency test in detail. The procedure is as follows:

"A solution of potassium silver cyanide is prepared by dissolving 10 grams of silver cyanide (AgCN) and 5 grams of '99%' potassium cyanide in a little water and diluting up to 1000 cc. It is then adjusted by addition of a little more KCN or AgCN until the solution indicates from 0.12% to 0.15% free KCN by titration with standard silver nitrate. The titration is best made by using a 10-cc. or 20-cc. sample, adding 1 cc. of 2% potassium iodide, and a slight excess of ammonia, as the end point is then sharper. Or 15 grams of pure crystallized KAg(CN) , may be dissolved in a liter of water and 1.5 grams KCN added.

"Weigh out 0.5 gram zinc dust into a 300-cc. beaker. Add a few cubic centimeters of water and stir until zinc is well mixed, then pour in 250 cc. of the prepared solution, stirring vigorously. See that all lumps are broken up, and continue stirring for fully five minutes. Stir occasionally (at least every 10 minutes) until the end of two hours from the addition of the solution. Then filter upon an 11-cm. filter, wash precipitate thoroughly, sprinkle with test lead, wrap it carefully in the paper, place in a scorifier with about 20 grams test lead, burn paper cautiously in muffle, scorify five minutes, cupel at low temperature and weigh silver. Milligrams silver obtained from 0.5 gram zinc $\times 0.0606$ = percentage precipitating efficiency."³

* Homestake Mining Co.

¹ *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, vol. xii, No. 8, p. 332 (Feb., 1912); reprinted in *Engineering and Mining Journal*, vol. xciii, No. 19, p. 943 (May 11, 1912).

² *Engineering and Mining Journal*, vol. xcv, No. 16, p. 793 (Apr. 19, 1913).

³ W. J. Sharwood, *loc. cit.*

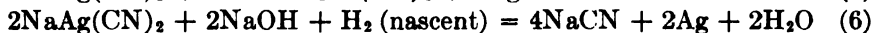
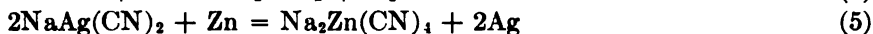
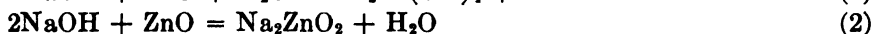
Recently I substituted the equivalent of sodium cyanide for the potassium cyanide, with no variation in the results.

A test was made with a gold solution that was equivalent to the standard silver solution. The efficiency shown was one-third to one-half that with the silver solution, in different trials. In one set of experiments a mixture of these gold and silver solutions in equal parts was used. The beads obtained were parted, and were found to contain about four atoms of silver to one of gold. The calculated efficiency was intermediate between those obtained with gold alone and silver alone, being somewhat higher than the average of the two values. This agrees with the fact that in practice gold is more difficult to precipitate than silver.

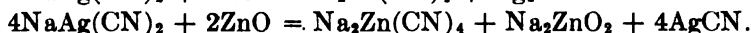
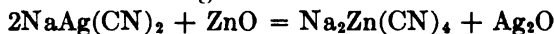
The following observations were made in the course of my experiments: Slight variations in the free cyanide present in the precipitating solution affect the results very little. However, if the free cyanide is reduced materially, to about 0.01 per cent., the results are very different. Tests agitated for short periods indicate that the rate of precipitation is much slower. In some cases exposure of the zinc to moist air for several hours so as to increase the oxide present seemed to increase the efficiency, although the amount of metal present in 0.5 g. was less. Also, in some cases, the addition of a little freshly ignited zinc oxide to the solution at the start of the precipitation increased the amount of silver precipitated. This seems to be due to the action of the oxide on carbonates in the solution. In this solution of low cyanide strength there is not enough free cyanide to dissolve all the zinc oxide, and the excess of zinc oxide would react with the alkali carbonates introduced from the cyanide used, forming zinc carbonate and caustic alkali. It is well known that the presence of caustic alkali in a solution increases the rapidity and completeness of precipitation.

As the solutions in practice are ordinarily free from carbonates, containing their alkali as lime, the standard method of preparing the silver solution was altered as follows: The solution is made up to double strength in silver. After adjusting the cyanide strength to about 0.2 per cent. NaCN, a slight excess of lime water is added to convert carbonates into hydroxides, and the solution filtered. It is then made up to the final volume, and tested for protective alkalinity. If this is not equivalent to about 0.01 to 0.015 per cent. NaOH, the solution is adjusted by the addition of a little acid. I have never found the alkalinity lower than desired, due to the alkali present in the cyanide as impurity. Although this adds to the work of preparing the solution, it is worth the trouble, as the results are more uniform, and come nearer to actual practice. Slight variations in the solution, such as the addition of neutral salts and changes in temperature, have much less effect than with the old standard solution.

The following reactions may occur:



The following two do not occur:



For all practical purposes, reaction (5) may be taken as the important one, whether it takes place by itself, or with (3), (4), and (6) as intermediate stages. Reactions (1) and (2) consume cyanide and alkali, interfering with (3) and (4), and thus indirectly interfering with (6). The free cyanide and alkali in the standard solution are sufficient to take care of more oxide than is usually found in zinc dust.

As precipitation is always better in the presence of reasonable quantities of cyanide and alkali, and is electrolytic in its nature, the addition of neutral salts as noted above would affect the solutions of lower cyanide and alkali concentrations much more than the standard solution. It is also well known that chemically pure zinc is very much less active than zinc containing a reasonable amount of metallic impurities. I will touch briefly on the common impurities.

Impurities in Zinc Dust

Zinc Oxide.—Zinc oxide is always present, varying from under 5 per cent. to over 20 per cent. Unless zinc dust containing over 15 per cent. of the oxide is unusually efficient for other reasons, it should be condemned, as the oxide only dilutes it and consumes cyanide. The oxide can rarely be seen in any quantity, even under the microscope, but the indications are that it occurs as a film on the surface of the particles, or in minute pits on the surface. It is present, sometimes in considerable quantities, in the small amount of coarse material, as well as in the finest part of the dust, so it cannot occur as separate grains disseminated through the mass of the zinc.

In regard to the oxidation of zinc dust during handling, in dry air it is practically negligible. In some tests, exposing a layer about $\frac{1}{2}$ in. deep to dry air, with occasional stirring, for several weeks, the oxide was increased only a very few per cent. On the other hand, in moisture-saturated air the oxidation may be very rapid, amounting to as much as 2 or 3 per cent. per day, but the rate falls off as the oxidation goes on, presumably due to the protection of the particles by a film of oxide. Exposure of a fairly large surface, however, for a reasonable time, does

not reduce the efficiency of the zinc as much as is commonly supposed, as only the surface layer would be oxidized, and to only a small extent. In a few cases exposure for a few hours seems to be of advantage, as will be explained later.

Metallic Impurities.—The only metallic impurities of any importance are lead and cadmium. In the samples I have examined, the lead varies from almost none to about 5 per cent., and cadmium from none to about 4 per cent. Both are probably present as metal alloyed with the zinc, as they are less easily oxidized than zinc. I know of no simple way to prove this point, but it seems self-evident in the case of lead. In the case of cadmium, a rapid extraction with the ammonia-ammonium chloride solution used for dissolving zinc oxide dissolves no cadmium, but metallic zinc would precipitate cadmium from this solution, if any were dissolved.

The nature of alloys of cadmium and lead with zinc has not been studied by modern metallographic methods, but the following statements can be made: It is known that lead is slightly soluble in zinc, and on cooling, about 1.3 per cent. of lead is left in the zinc, as a solid solution, any more lead segregating as a solution of zinc in lead. Therefore, if cadmium is absent, the metallic portion of the zinc may contain 1.3 per cent. of lead in solution, possibly in molecular form, while any excess must be in larger particles. This can be shown by preparing an alloy, and cooling it slowly, cutting out all parts showing segregations of lead, and analyzing the remainder. Cadmium is soluble in melted zinc in all proportions, and insoluble in solid zinc; but, separating only at the instant of solidification, does not segregate in large masses. Lead is soluble in melted cadmium in all proportions, and partly soluble in solid cadmium, the excess of lead not segregating as in zinc. The probability is that cadmium would increase the solubility of lead in solid as well as melted zinc, and would certainly diminish the tendency toward segregation. I prepared an alloy containing about 2 per cent. cadmium and 2.2 per cent. lead, and let it cool slowly in a muffle. It seemed perfectly homogeneous, and if there was any undissolved lead, it was so finely disseminated as to be invisible on etching the specimen.

Lead is known to increase the activity of zinc in cyanide solutions, as it has a much lower electromotive force, and remains undissolved. It is not known whether the dissolved portion (in the zinc) is more efficient than segregations, or *vice versa*. Cadmium also has a lower potential than zinc, and would probably also help, but would not be as advantageous, as the difference in potential is less. From several analyses of precipitates, the indications are that the cadmium remains in the precipitate, which has been proved to be the case with lead. It is to be hoped that the metallography of this system will be worked out soon so as to answer the question of the distribution of the impurities. The following

analyses show the variation in some samples of zinc dust from different sources:

Lead	Cadmium	Zinc Oxide	Total	Per Cent. of Metallic Portion	
				Lead	Cadmium
0.15	0.00	8.3	8.45	0.16	0.00
1.18	3.1	9.6	13.9	1.31	3.43
1.75	1.7	8.45	11.9	1.91	1.86
1.80	1.2	17.85	20.85	2.19	1.46
1.82	1.85	10.15	13.8	2.02	2.06
2.04	2.15	11.65	15.85	2.31	2.46
2.06	3.9	10.5	16.45	2.30	4.36
2.38	1.85	11.85	16.1	2.70	2.10
2.60	1.25	6.3	10.15	2.78	1.34
3.05	1.9	16.15	21.1	3.64	2.27
3.65	1.15	10.6	15.4	4.08	1.29
3.73	2.45	12.2	18.4	4.25	2.79
5.20	1.7	7.25	14.15	5.61	1.83

In calculating the columns headed "per cent. of metallic portion," it was assumed that 100 per cent. less the zinc oxide was the metallic part. The error, if any, would tend to make the figures as calculated lower than the truth. The table shows that practically any combination within the limits given for lead and cadmium may be expected to occur.

Other Impurities.—Other metallic impurities are negligible. Of the non-metallic impurities, nearly all samples contain a small amount of insoluble material, generally sand or fireclay; the amount is very small, and the only effect can be to dilute the dust with a trace of inert matter. Some zinc dust also contains a small amount of carbonaceous matter, such as coal dust, very seldom more than a small fraction of 1 per cent. This is also inert. This was demonstrated by intimately mixing 0.5 g. of zinc dust with 10 per cent. of finely ground coke dust in one case, and the same amount of graphite in another trial. In neither case did the amount of silver precipitated differ materially from that obtained when using the same amount of zinc dust by itself.

Another impurity that may occur is carbon dioxide, combined with the zinc determined as oxide. If the dust has been exposed to moist air, it is reasonable to assume that some zinc carbonate would be formed. As the solution being precipitated contains lime, the carbonate would cause the precipitation of calcium carbonate on the zinc grains, preventing efficient contact with the solution. All precipitates contain some calcium carbonate, but ordinarily, if there has not been much chance for the access of air to the solution, the amount is so small as to make the effect of carbon dioxide in the zinc dust very slight.

There is one other material that is found in many lots of zinc dust that may be of some importance. This material is oil of some sort. If present, it may be detected by extraction with ether. I do not know how it is introduced into the dust. It may be deposited in the cooler part of the condenser, and come from the reducing agent used, or it may be added later to diminish the tendency toward oxidation. On dissolving oily zinc dust in dilute acid, the action is very slow, due in part to the protection by a film of oil, and also to an effect similar to flotation methods of concentration, the dust being floated in a rather compact mass. An oil-free dust can be oiled by moistening with a solution of paraffin or some heavy oil in ether, and evaporating the solvent, and then it acts like the oily dust. It is the dust containing appreciable quantities of oil that seems to be benefited by some exposure to air; possibly the formation of a film of oxide removes the oil from the surface and leaves clean metal for precipitation.

Screen Analyses of Zinc Dust

Ordinarily from less than 1 per cent. up to nearly 5 per cent. will remain on a 200-mesh screen, and from one-fourth to three-fourths of this will remain on a 100-mesh screen. The coarse particles are partly sand, bits of fireclay, and similar things; usually there are some coarse particles of zinc. The amount that passes through a 200-mesh screen does not indicate the real fineness of the zinc dust, as much depends on the average size of the through-200 portion. A further separation may be made by elutriation.⁴ Much may depend upon the composition of the various fractions.

In some cases lead determinations were run on the separate fractions. In one lot the fine portions contained progressively less lead, and in another contained more than the coarser fractions. Cadmium determinations were not made on these fractions. As a rule, the fine portions contain higher percentages of zinc oxide, and there may be enough to outweigh the effect of a high proportion of fines. In other lots, however, the zinc oxide was fairly evenly distributed through all the sizes. It may be that determinations of metallic zinc in the fractions would be of value. The difficulty in determinations of this sort is the chance for oxidation during the manipulation, especially the drying of the fractions.

Substitutes for Zinc Dust

There have been several zinc products proposed as substitutes for dust, but none of those so far suggested have been of any consequence. They have been mechanically made, of the nature of chips or cuttings;

⁴ W. J. Sharwood, *loc. cit.*

or they have been similar in character to test lead, of varying degrees of coarseness; some have been finely crystalline. All that I have seen have the same failing of coarseness. The efficiency of various substitutes ranges from 5 to 20 per cent. of the efficiency of an average dust, by the silver-precipitation method.

Conclusions

Zinc dust satisfactory for precipitation of precious metals from cyanide solutions should satisfy the following requirements: It should be fine; that is, most of it should pass through a 200-mesh screen, and the fine portion should be very much finer than the screen opening. As the surface exposed per unit weight varies inversely as the diameter of the particles, and the number of grains varies inversely as the cube of the diameter, if the average diameter of one lot of dust is one-half that of another, the finer would have twice the surface and eight times the number of particles. The dust would remain in suspension better, and, if uniformly distributed, the distance between grains would be half that of the coarser material.

The oxide should be low, but very low oxide does not necessarily indicate a good zinc. There should be a reasonable amount of lead, over 1 per cent. As high as 4 per cent. may do no harm. Cadmium is usually present, and some cadmium may take the place of part of the lead. If oil is present, it should not be in such quantity as to prevent good mixing of the dust with the solution. Precipitation with a standard solution of silver cyanide gives a rapid method for approximating the value of a particular lot of zinc dust. If the efficiency is well over 40 per cent. (100 per cent. being the theory for complete replacement of pure zinc by silver), the zinc is satisfactory. If under 30 per cent., the dust will probably give poor results.

The condition of the metallic impurities may have much influence. It is quite possible that the composition of different particles may vary according to the time in the distillation when they were condensed, or because of mixing of lots of dust of decidedly different composition. In a case of this sort, the precipitation test is the only simple method of determining what may be expected in actual work.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Radiography of Metals

BY WHEELER P. DAVEY,* SCHENECTADY, N. Y.

(San Francisco Meeting, September, 1915)

In an article in the *General Electric Review*, January, 1915, reference was made to the X-ray examination of a steel casting $\frac{9}{16}$ in. thick. Fig. 1 shows one of the radiographs thus obtained. All these radiographs showed plainly the tool marks on the surface of the casting. All but one showed peculiar markings the shape of which strongly suggested that they were indeed the pictures of holes in the interior. A circular piece, 1 in. in diameter, was punched from the casting at a point where the radiograph shown in Fig. 1 indicated that a blow-hole should be found. (Location of sample shown by circle in Fig. 1.) Fig. 2 is a photograph of the end of the punching, showing the hole that was found. Since that article was written it has seemed desirable: (1) to obtain data from which the exposure necessary for any thickness of steel could be at once calculated; (2) to find the thickness of the smallest air inclusion which could be radiographed in a given thickness of steel; (3) to determine in what direction to hope for further progress; and (4) to develop the technique of radiographing metals.

In order to gain some preliminary data, several pieces of $\frac{1}{2}$ -in. (12.5-mm.) boiler plate were obtained, 5 by 7 in. (12.5 by 17.5 cm.) in size. In one of these, holes were drilled in such a way that the axis of each hole was midway between the faces of the steel and parallel to those faces. The diameters of these holes were as follows:

Hole number	Diameter
1.....	$\frac{1}{4}$ in. = 6.3 mm.
2.....	$\frac{1}{8}$ in. = 3.1 mm.
3.....	$\frac{1}{16}$ in. = 1.6 mm.
4.....	$\frac{1}{32}$ in. = 0.8 mm.
5.....	$\frac{1}{64}$ in. = 0.4 mm.

Exposures were made on Seed X-ray plates at 20 in. (50 cm.) distance with Coolidge tube X-117, operated on Scheidel-Western induction coil with mercury turbine break. The X-ray plate was placed on a sheet of $\frac{1}{8}$ -in. (3.1-mm.) lead. The steel was placed above this and a lead

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FIG. 1.—RADIOGRAPH OF STEEL CASTING, $\frac{9}{16}$ IN. THICK, SHOWING FLAW IN INTERIOR.

cover was placed over the whole in such a manner that the cover and backing made a complete lead shield for the X-ray plate. (See Fig. 3.) A rectangular hole in the cover allowed such X-rays as were able to penetrate the steel to reach the X-ray plate. This afforded complete protection



FIG. 2.—BUTTON, 1 IN. IN DIAMETER, CUT FROM STEEL CASTING.

against secondary rays. Without such precautions, the effect on the X-ray plate of secondary rays would have been greater than that of the rays used to take the picture. If the steel had been 2 or 3 ft. (60 or 90 cm.) square, such precautions would have been unnecessary. By placing the pieces of boiler plate on top of one another, any thickness of steel de-

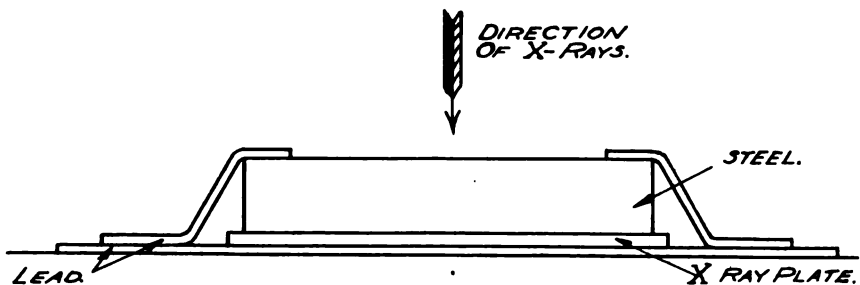


FIG. 3.

sired could be obtained. Exposures were made at 11, 13, and 15 in. (27.5, 32.5, 37.5 cm.) parallel spark gap between points. An attempt was made to use a 17-in. (42.5-cm.) spark gap, but was abandoned on account of flashing in the tube.

Results are tabulated below:

Thickness of Steel	Plate	Spark Gap		Exposure in MA-Min.	Holes Visible
		In.	Cm.		
$\frac{1}{2}$ in. (12.5 mm.)	D	11	27.5	7	1-2-3-4-5
	A	13	32.5	4	1-2-3-4-5
	B	15	37.5	2	1-2-3-4-5
1 in. (25 mm.)	E	11	27.5	45	1-2-3-4-5
	F	13	32.5	19	1-2-3-4-5
	G	15	37.5	10	1-2-3-4-5
$1\frac{1}{2}$ in. (37.5 mm.)	H	11	27.5	45	1-2-3 very faint
	I	13	32.5	30	1-2-3 very faint
	K	13	32.5	90	1-2-3 faint
	J	15	37.5	30	1-2-3-4-5 very faint
	L	15	37.5	60	1-2-3-4-5 faint

This really means, of course, that at 13-in. (32.5-cm.) spark gap 90 milliamperes-minutes is sufficient to enable one to notice the difference in blackening between exposures through $1\frac{1}{16}$ and $1\frac{1}{2}$ in. (36 and 37.5 mm.), but is not sufficient to enable one to detect the difference in blackening between exposures through $1\frac{15}{32}$ and $1\frac{1}{2}$ in. (36.8 and 37.5 mm.) of steel.

The above results were necessarily incomplete, since the plates were by no means all of the same density. They served, however, to show two things:

1. With the voltages which can now be used, it is impracticable to radiograph through more than $1\frac{1}{2}$ in. (37.5 mm.) of steel with tungsten target tubes because of the time required.

2. The use of high voltages does not seem to reduce appreciably the clearness of the picture obtained. (It was to have been expected from the published data on scattering in aluminum that enough scattered radiation would have been produced to blur the pictures, but plate B apparently shows as good detail as does Plate D.)

It remained to confirm the above conclusions with data of a quantitative nature. Seed X-ray plates were therefore exposed under the same conditions as before, except that none of the slabs of steel used had been drilled. For each thickness of steel, all the exposures at a given spark gap were made on the same plate. Each plate, then, showed a series of steps which increased in density from one end of the plate to the other. Thickness of steel, spark gap, and milliamperes-minutes were recorded on each plate by means of lead numbers. The following plates were thus exposed:

Plate	Thickness of Steel		Spark Gap	
	In.	Mm.	In.	Cm.
No.				
216	$\frac{1}{2}$	12.5	11	27.5
217	$\frac{1}{2}$	12.5	13	32.5
218	$\frac{1}{2}$	12.5	15	37.5
221	1	25.0	11	27.5
220	1	25.0	13	32.5
219	1	25.0	15	37.5
222	$1\frac{1}{2}$	37.5	15	37.5

A study of these plates showed the following facts:

Let $E_{\frac{1}{2}}$ be the exposure in milliampere-minutes necessary to produce a given darkening of the plate through $\frac{1}{2}$ in. (12.5 mm.) of steel, and let E_1 and $E_{1\frac{1}{2}}$ be the exposures necessary to produce the same darkening through 1 in. (25 mm.) and $1\frac{1}{2}$ in. (37.5 mm.) respectively.

Then at 11-in. (27.5 cm.) gap, $E_{\frac{1}{2}} : E_1 = 1 : 11$.

At 13-in. (32.5 cm.) gap, $E_{\frac{1}{2}} : E_1 = 1 : 8$.

At 15-in. (37.5 cm.) gap, $E_{\frac{1}{2}} : E_1 = E_1 : E_{1\frac{1}{2}} = 1 : 8$.

Also, through both $\frac{1}{2}$ in. (12.5 mm.) and 1 in. (25 mm.) of steel,

$$E_{13\text{-in. (32.5 cm.) gap}} : E_{11\text{-in. (27.5 cm.) gap}} = 1 : 4$$

and

$$E_{15\text{-in. (37.5 cm.) gap}} : E_{13\text{-in. (32.5 cm.) gap}} = 2 : 3.$$

It is at once evident that *either* X-rays from a tungsten target at 13-in. (32.5-cm.) gap are more penetrating than when produced at 11-in. (27.5-cm.) gap, *or* the X-ray plates used are more sensitive to the rays produced at 13-in. (32.5-cm.) gap. There is other evidence to show that the first of these conclusions is the more probable, but it is the *effect of the X-rays on the plate* which is of prime importance in this work, so that from a radiographic standpoint we may say in any case that the *effective penetration* of the rays is a little greater at 13-in. (32.5-cm.) than at 11-in. (27.5-cm.) gap.

In the same way we may conclude that the effective penetration at 15-in. (37.5-cm.) is the same as at 13-in. (32.5-cm.) gap. There is, however, a marked decrease in the amount of exposure required as the voltage across the tube (as measured by the spark gap) is increased. This may be due to one of two causes: *either* the efficiency of transformation from the kinetic energy of the cathode stream to the energy of the X-rays may be greater at the higher voltage, *or* there may be some peculiarity in the wave-form produced by the induction coil such that a great deal of energy is given off at a voltage corresponding to a 13-in. (32.5-cm.) gap when the coil is operated so as to give a maximum voltage corresponding to a 15-in. (37.5-cm.) gap.

From the data at hand it is easily possible by well-known means to construct formulas for computing the exposure necessary for radiographing steel at various spark gaps.

Let Q_0 be the quantity of X-rays impinging on the steel during the exposure.

Let Q be the quantity of the rays which pass through the steel.

Let x be the thickness of the steel.

Let λ be the coefficient of absorption, and

Let e be the base of natural logarithms.

Then if the X-rays are homogeneous,

$$Q = Q_0 e^{-\lambda x}$$

Now at 15-in. (37.5-cm.) gap we know that $\frac{E_{1/2}}{E_1} = \frac{E_1}{E_{1 1/2}} = \frac{1}{8}$.

The rays given off at 15-in. (37.5-cm.) gap are therefore practically homogeneous. Since $\frac{E_{1/2}}{E_1} = \frac{1}{8}$ at 13-in. (32.5-cm.) gap, we may assume that these rays are also practically homogeneous. Rays given off at 11-in. (27.5-cm.) gap are still sufficiently homogeneous, after having passed through the first few hundredths of an inch of steel, to permit their being treated as though they were actually homogeneous. Calculations for exposures at 11-in. (27.5-cm.) gap are to be considered as only good approximations.

For 15-in. (37.5-cm.) gap we have

$$Q/Q_0 = \frac{1}{8} = e^{-\lambda x} = e^{-\frac{1}{2}\lambda}$$

$$\log 8 = \frac{1}{2}\lambda = 2.079$$

$$\lambda = 4.16 \text{ in.}^{-1} = 1.64 \text{ cm.}^{-1}$$

Likewise for 13-in. (32.5-cm.) gap

$$\lambda = 4.16 \text{ in.}^{-1} = 1.64 \text{ cm.}^{-1}$$

Applying the same method for 11-in. (27.5-cm.) gap

$$\lambda = 4.80 \text{ in.}^{-1} = 1.89 \text{ cm.}^{-1}$$

Now at 15-in. (37.5-cm.) gap and 20 in. (50 cm.) distance 0.8 milli-ampere-minutes gives a good exposure through $\frac{1}{2}$ in. (12.5 mm.) of steel. A corresponding darkening would have been produced on a bare plate by an exposure of 0.1 milliampere-minutes. This corresponds to Q in our formula. We may therefore write, since $Q_0 = E$,

$$0.1 = E \cdot e^{-4.16x}$$

$$10E = e^{4.16x}$$

$$\log_e 10E = 4.16x$$

$$\log_{10} 10E = 1.80x$$

$$E = \frac{1}{10} \log_{10}^{-1} 1.80x.$$

where x is the thickness of the steel in inches

$$\text{or} \quad E = \frac{1}{10} \log_{10}^{-1} 0.71x$$

where x is the thickness of the steel in centimeters.

The corresponding formulas for 13-in. (32.5-cm.) gap are

$$E = \frac{3}{20} \log_{10}^{-1} 1.80x \quad (x \text{ in inches})$$

$$E = \frac{3}{20} \log_{10}^{-1} 0.71x \quad (x \text{ in centimeters}).$$

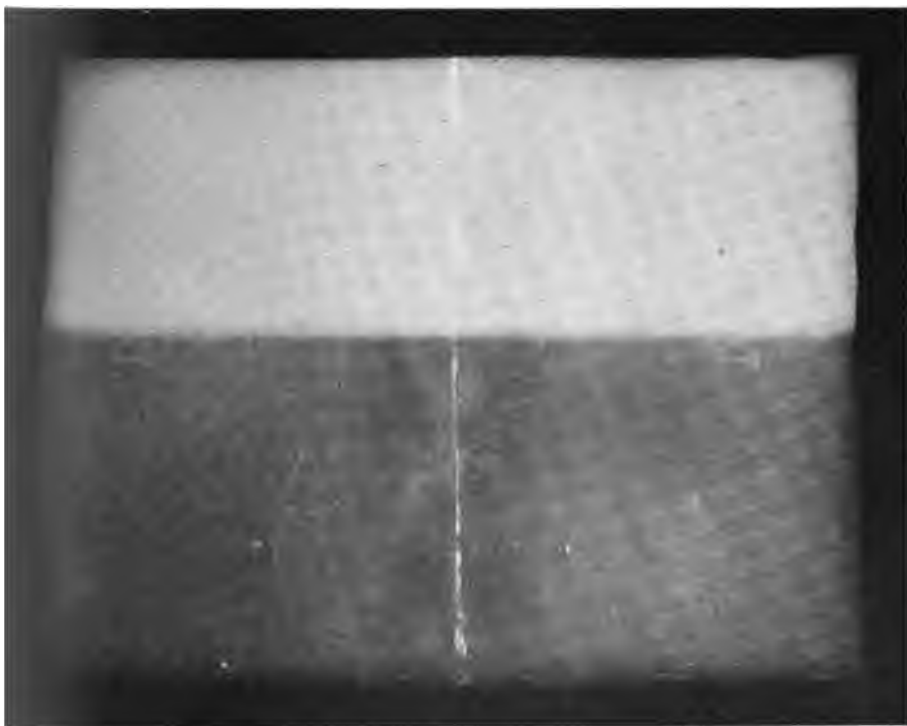


FIG. 4A.



FIG. 4B.

The approximate formulas for 11-in. (27.5-cm.) gap are

$$E = \frac{3}{5} \log_{10}^{-1} 2.09x \quad (x \text{ in inches})$$

$$E = \frac{3}{5} \log_{10}^{-1} 0.82x \quad (x \text{ in centimeters}).$$

It remained to find the thickness of the smallest air-inclusion which could be radiographed in steel at 15-in. gap. For this purpose two plates

of steel were taken. The faces were machined flat and in one of them a slot was cut, thus giving a *wedge* of air. The slot and the faces of the steel plates were then ground smooth. When completed, each plate was $\frac{5}{8}$ in. (15.5 mm.) thick. The air wedge was 10 in. (250 mm.) long, 1 in. (25 mm.) wide and $\frac{9}{64}$ in. (3.52 mm.) thick at its thick end. When the two plates were bolted together, the air wedge simulated a blow-hole in a



FIG. 5A.



FIG. 5B.

casting. The wedge was then radiographed at 15-in. (37.5-cm.) gap. When the X-ray plates were dry the place was noted at which the outline of the wedge was barely visible. In order to avoid error, only a small portion of the wedge was viewed at one time, the rest being blocked off with cardboard. It was found that an air inclusion 0.021 in. (0.52 mm.) thick could be detected in $1\frac{1}{4}$ in. (31.3 mm.) of steel. In $\frac{5}{8}$ in. (15.6 mm.), an air inclusion of 0.007 in. (0.18 mm.) could be detected.

Besides the work above outlined, a great deal of work has been done in the actual taking of pictures, so that the technique of radiography through metals might be worked out.

A record of a single example will suffice. Four samples of autogenous welds in steel were obtained. The welding had been done with an oxy-acetylene flame. The samples were $\frac{1}{2}$ in. (12.5 mm.) thick and

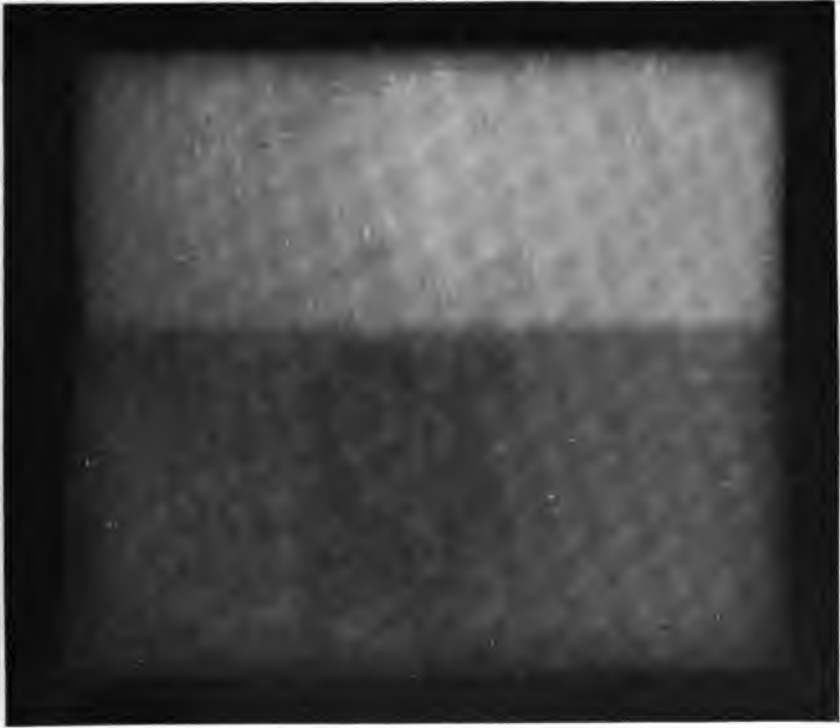


FIG. 6A.



FIG. 6B.

about 4 in. (10 cm.) square. The faces were fairly rough. Sample No. 1 had only been welded on the surfaces. (See Fig. 4B.) Sample No. 2 had been insufficiently heated so that there was incomplete fusion of the metal at the center. (See Fig. 5B.) In welding sample No. 3 an excess of oxygen had been used in the flame, causing the presence of oxide on the surface. (See Fig. 6B.) Sample No. 4 was considered to be a good weld. (See Fig. 7B.) One-half of each face was machined off, so that half the

length of the weld was between flat, parallel faces; the other half was left under the original rough surfaces. As a result, one-half of each sample was $\frac{1}{2}$ in. (12.5 mm.) thick and the other half was about $\frac{3}{8}$ in. (9.4 mm.) thick. Radiographs were taken at 15-in. (37.5-cm.) gap under the conditions described above. Reference to the formula for exposure at 15-in. (37.5-cm.) gap shows that the exposures through the $\frac{1}{2}$ -in. (12.5-mm.) and

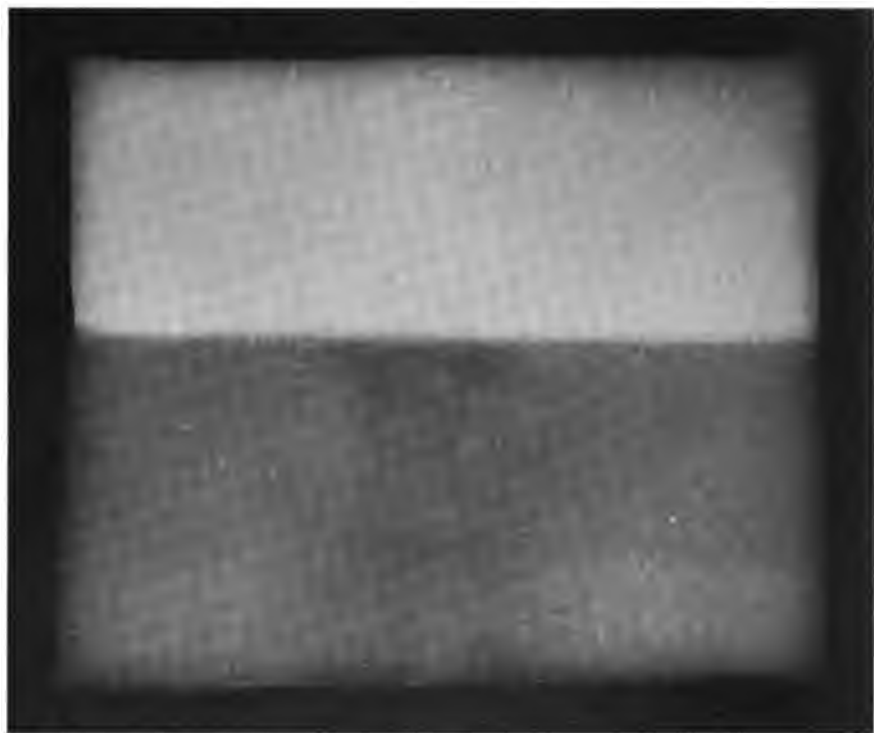


FIG. 7A.



FIG. 7B.

$\frac{3}{8}$ -in. (9.4-mm.) portions were in the ratio of 1 to 1.7. The resulting radiographs are shown in Figs. 4A, 5A, 6A, and 7A.

Fig. 4A shows clearly the unwelded center of sample 1 in both portions of the picture. Fig. 5A shows, in both portions of sample 2, the holes caused by the metal not having been thoroughly fused at the center. That portion of Fig. 6A which was taken through the machined end of the weld of sample 3 would seem to indicate a porous structure. Such a

structure was evident during the machining. The portion of the picture taken through the unmachined end of the weld did not show such a structure with certainty. This was to have been expected, as the inequalities in thickness due to the uneven surface were at least as great as those due to porous or frothy structure. Fig. 7A shows that as far as gross structure is concerned, sample 4 was a good weld.

It is of course self-evident that a radiograph gives only the gross structure of the metal, and no information as to the "grain," crystal interlocking at the edge of the weld, etc. A radiograph does, however, give valuable information as to the presence of blowholes, slag inclusions, porous spots and defects of like nature, which could not be found otherwise except by cutting into the metal. Unfortunately, no fluoroscopic screen now known is sensitive enough for this work; all work in metals must be done radiographically. An inspection of the formulas derived above makes one feel that, for the present at least, radiography of steel is a commercial possibility only up to thicknesses of $\frac{1}{2}$ in. (12.5 mm.). For greater thicknesses, the time required is excessive. The great saving in time which is gained by the use of a 15-in. (37.5-cm.) instead of a 13-in. (32.5-cm.) spark gap makes it seem probable that a further increase in the voltage across the tube would allow us to radiograph still greater thicknesses of steel.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Petroleum as Fuel under Boilers and in Furnaces for Heating, Melting, and Heat Treatment of Metals

BY W. N. BEST, F. R. S. A., NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

INTRODUCTION

CRUDE oil attracted attention because of its excellence as a fuel for open-hearth furnaces; for making crucible steel and brass; for melting copper, lead, tin, zinc, nickel, silver, malleable iron, gray iron; and for the production of steam in all sizes and types of boilers. For heating furnaces, mold-drying ovens, core ovens, ore roasters, calciners, hot-air furnaces, sand drying, asphalt road work, etc., its superiority over other fuels is due to the fact that the heat is at all times under perfect control, so that a constant temperature may be attained and maintained at the will of the operator. This means a maximum amount and uniform quality of work.

Crude oil is of two kinds, paraffine base and asphaltum base, and varies from 11 to 46 gravity Baumé. The light-gravity oil does not require heating before use; but oil of less than 20 gravity Baumé should be heated to just below its vaporizing point, in order to reduce its viscosity and to obtain the highest efficiency of combustion. Heavy California oil vaporizes at 230° F., Mexican crude oil at from 175° to 210° F., and Beaumont, Texas, oil at 142° F. Fuel oil vaporizes at 130° F.

Scientific Installation Essential

The burning of oil is an art based on science, and the "rule of thumb" should never be employed, especially in heating heavy crude oil. Thermometers should be used on both suction and supply pipes so that the fireman or smith will at all times know the temperature of the fuel. The indicators on such thermometers should be large enough to be readily seen by the operator, night or day.

It is necessary to have an unvarying pressure of steam or air through the burner, to atomize the fuel. Also, the air used to support combustion should be delivered through a nozzle under the burner at a constant pressure. The oil pressure must not fluctuate; under no circumstances should it vary more than 1 lb. The oil pump, which should be brass lined, with the aid of the pulsometer and superheater, will insure a constant circulation of fuel to the various burners, maintaining an unvarying pres-

sure on the supply pipes. All oil pipes should be of adequate size and carefully laid.

A non-carbonizing burner of modern construction, that will externally atomize the fuel and distribute the heat over the entire width and length of the furnace or firebox, should be used. For a furnace, the use of only one burner giving a flat flame covering the entire hearth is always pref-



FIG. 1.—HIGH-PRESSURE OIL BURNER MOUNTED FOR MARINE OR STATIONARY BOILERS BURNING OIL OR TAR EXCLUSIVELY AS FUEL.

erable. If steam is used through the burner to atomize the oil, it should be taken from the top or dome of the boiler or superheater so that it will be as dry as possible. The atomizer orifice in the burner should be as small as practicable to provide for the minimum and maximum capacity of the burner.



FIG. 2.—LOCOMOTIVE BURNER.

Combustion Chamber

The function of a combustion chamber is to provide space in which the atomized fuel may unite with sufficient air for combustion before it reaches the furnace proper. In the more modern furnace practice only a sufficient amount of steam or air is forced through the burner to atomize and distribute the oil, while the remainder (ordinarily about $1\frac{9}{20}$) of the air necessary for combustion is delivered at from 2 to 3 oz. pressure

through a nozzle, often located below the burner, and provided with a sliding gate, so that the quantity of air entering the combustion chamber is readily controlled by the operator. It is necessary that the flame made by the burner shall fit the combustion chamber perfectly, so that the volume air delivered into the chamber will pass through the flame. When this rule is observed, the melter cannot oxidize the metal, and injurious effects from sulphur in the fuel are prevented. The proportions of the chamber must be adequate to the size of the furnace and the temperature maintained therein.

BOILER SERVICE

The use of coal involves the handling of coal and ashes, the employment of additional firemen, etc., all of which is avoided by the use of oil. When, in view of this saving, the price of oil compares favorably with that of coal, the former is highly preferable on other grounds also. It is clean, simple and cheap in operation; and since it requires accurate control, it utilizes and trains the judgment of the fireman, and thus elevates the character of the labor employed.

In the equipment of locomotive, marine, and stationary boilers, the first consideration is the protection of the metal constituting the elements of the boiler. This can be effectively accomplished at a small cost.

Locomotive Service

As soon as a locomotive is changed from coal to oil fuel (which can be done at a very small cost), the train-tonnage of the engine is increased 15 per cent., because the locomotive can easily carry the steam pressure at all times at just below its "popping-off" point. This, of course, cannot be done while using coal as fuel.

For locomotive service the most modern practice is "the duplex oil system," which employs two burners, a small and a large one. The former, used as the engine leaves the round-house and operated continuously thereafter, serves as a pilot light, as well as to keep just sufficient heat in the firebox to maintain the temperature and the steam required when the locomotive is standing still. It keeps the steam at just below the "popping-off" point when only the air pump is running, and no other work is being done. The large burner, ordinarily placed above the smaller one, is operated when the locomotive is at work. By this system the life of the boiler is increased, and the handling of the locomotive becomes much simpler. Gravity oil feed is ordinarily used in locomotive service. Air pressure should not be used on the locomotive oil tank to aid in forcing the fuel to the burner; but in stationary or marine practice 10 lb. pressure should be maintained on the oil-supply pipe.

Marine Boiler Service

In marine service oil, at the will of the fireman, readily increases the boiler capacity 50 per cent. or more. The fuel tanks on the vessel can be filled in a very few minutes, regardless of climatic conditions. There is no dust or dirt—a great advantage, since the dust from coal, settling in the state-rooms and around other parts of the vessel, is very annoying on shipboard. In an emergency, when the vessel must be run at maximum speed, liquid fuel quickly responds to the demand for extra work.

Power Plants

For stationary boilers, especially in electric-light plants, where there may be a battery of "standby" boilers, the use of oil permits the boiler capacity to be easily doubled in an incredibly short time, without injury to the elements of the boilers. One man can fire and water-tend twelve 300-hp. boilers.

COMMERCIAL FURNACES

Some years ago manufacturers often contracted for the complete installation of oil furnaces in their works, but they have found it better policy to have their own masons construct the furnaces, purchasing designs and oil burners, etc., from well-known engineers in this particular line. Under this system, greater care is exercised in the construction of the furnace, for the local mason knows that any errors or defects in workmanship will be brought to his attention, and he will be held responsible for them. Manufacturers have found, moreover, that their own masons can construct the furnaces for 50 per cent. less than any outside furnace manufacturer can afford to estimate. I do not here refer to open-hearth furnaces. They require special brick, and workmen skilled in this particular construction.

Firebrick

The selection of firebrick should be carefully made and only the best quality used; for it requires as much time and labor to lay a bad brick as a good one. The brick should withstand the temperature required without dripping, disintegrating, or spalling. They must not be laid in a layer of fireclay, as is the practice with red brick, but simply dipped in a thin mixture of fireclay and water, before being placed in position in the walls or arch of the furnace. The fireclay should be purchased from the firm manufacturing the firebrick. If a different fireclay is used, it will not adhere so well to the brick, and furthermore, it will not fuse or bond with the surface of the latter.

In drop-forge, annealing, or heat-treating furnaces for soft steel,

carbon steel, or high-speed steel, the firebrick used should show no perceptible expansion or contraction when heated to the required temperature. Specifications for such a quality of brick are as follows:

	Per Cent.
Silica.....	56.15
Alumina.....	33.295
Peroxide of iron.....	0.59
Lime.....	0.17
Magnesia.....	0.115
Water and inorganic matter.....	9.68

Old Method vs. New Method

In annealing furnaces or furnaces for the heat treatment of steel, the use of pyrometers is essential, since the value of steel depends upon its heat treatment.

A few years ago, we often heard the expression, "bring the furnace to a cherry red for annealing," or, "for case hardening, etc., bring the furnace to a bright red or yellow." "Heat the steel to dark red, quench it and draw it to an indigo color or a peacock blue," etc., etc. Those expressions are almost forgotten now, and recording pyrometers are used instead. Without these recorders the output is not satisfactory to the operator, manufacturer, or user of the steel. Daily operations have now become a matter of official record. I know of one firm which used to import water from a foreign country in which to quench its steel, as the workmen considered the water found in the United States inferior for tempering steel. It is pleasant to add that this plant has now adopted modern ideas of shop practice, and is not quenching its steel in water.

Special Furnace Designs

Furnaces should be specially designed for each class of work. In the United States a manufacturer largely makes a certain specialty, and if he decides to introduce oil furnaces by reason of their generally reported merits—increased output, improved quality of product, etc.—he should take great care to get the size and form of furnace best adapted for heating his stock. For example, A has stock to be heated of the same size as his neighbor B; and yet A and B may require furnaces of different sizes, for the reason that A requires three times as long as B to bring his forgings to shape and size. B's furnace must have a charging space three times as large as A's, or else B's smith must wait for his stock to be heated instead of the metal being hot and ready for the man. Again, should A have a furnace of the same large size as B, his metal would be constantly waiting on the operator, which would result in considerable scaling of the charge, and a loss in metal and fuel. An engineer, going through the works of

various manufacturers, sees hundreds of such misfits, resulting in poor shop practice. We all agree that in the forging of iron and steel the metal should be removed from the furnace as soon as heated to the proper forging temperature, and at the same time the furnace must be of ample capacity to heat sufficient stock so that the smith will not have to wait for the metal to be heated.

Modern Smith-Shop Practice

In modern drop-forge shops and small heat-treating plants, the best practice is to have the furnaces so constructed as to be portable; that is, incased in a substantial shell and provided with lifting irons so that they can be easily handled by the shop crane. These furnaces should rest on two or more concrete columns, so that the charging space will be at a convenient height for the operator.

When the refractory lining has been burned out, the furnace is removed by the cranes to the mason's room, in which it is relined by a shop mason, while another furnace is substituted. In this way the hammers or tools do not remain idle, and the operator loses no time. These changes are made during the night at the smallest possible expense, a stock of relined furnaces being kept in the supply room ready for use.

The mason is provided with a work room and blue prints, which enable him to make the particular form of refractory construction required for the type of furnace to be relined.

For drop-forge furnaces the use of magnesite bottoms is becoming very popular, because magnesite has no affinity for the hot metal, while clay brick or sand often adheres to the forging, which is very annoying to the smith.

Heat Treatment of Steel

As already remarked, the value of steel depends wholly upon the heat treatment which it receives. To obtain the desired results, it is essential to establish and maintain an even temperature throughout the entire length and width of the furnace. For the heat treatment of carbon steel, which requires an indirect-fired furnace, this can only be done by means of graduated heat ports. Only one burner should be used, the heat therefrom passing from the fire chamber into the charging space of the furnace through graduated heat ports substantially as shown in the accompanying Fig. 3. The size and location of these heat ports is an engineering problem requiring careful consideration; for if they are not scientifically and accurately proportioned the incoming air used for the atomization of the fuel or to support combustion will cause an excessive heat at the end of the furnace opposite the burner.

For high-speed tool steel a direct-fired furnace is necessary; and the

more modern types have a preheating chamber above the charging space. The waste gases from the lower chamber passing up into the preheating chamber, slowly preheat the charge before it is passed into the furnace proper, thus preventing the too sudden expansion of the metal. (See Fig. 4.)

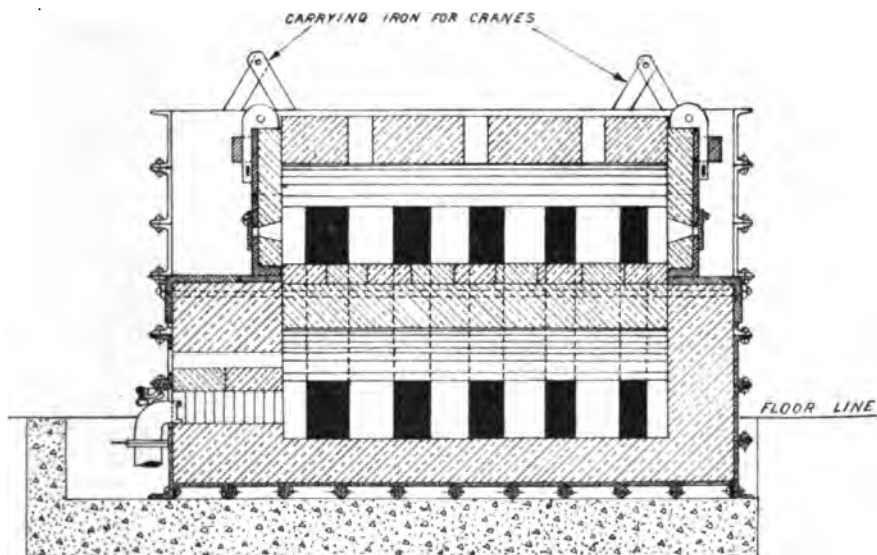


FIG. 3.—INDIRECT-FIRED FURNACE.

Brass Furnaces

The most modern practice is to charge the brass gates, old castings, etc., into a 2-ton or 4-ton melting furnace. From this furnace the metal is poured into ingots. These are tagged with the number of the heat and carefully analyzed by the metallurgist. When required these ingots are charged into a crucible furnace together with the required amount of copper, tin, or other alloy needed to meet the specifications of the purchaser.

A few years ago direct-fired furnaces were used in many brass foundries; but this method has been discarded as it resulted in an excessive loss in metal, while with oil-fired crucible furnaces the loss does not exceed that attending the use of coke- or coal-fired crucible furnaces.

Plate Heating

A localized heat is detrimental in a plate-heating furnace such as is used in boiler shops. Oil is an ideal fuel for this class of work since it insures an absolutely even heat throughout the entire charging space of the furnace, as well as a larger output than any other fuel.

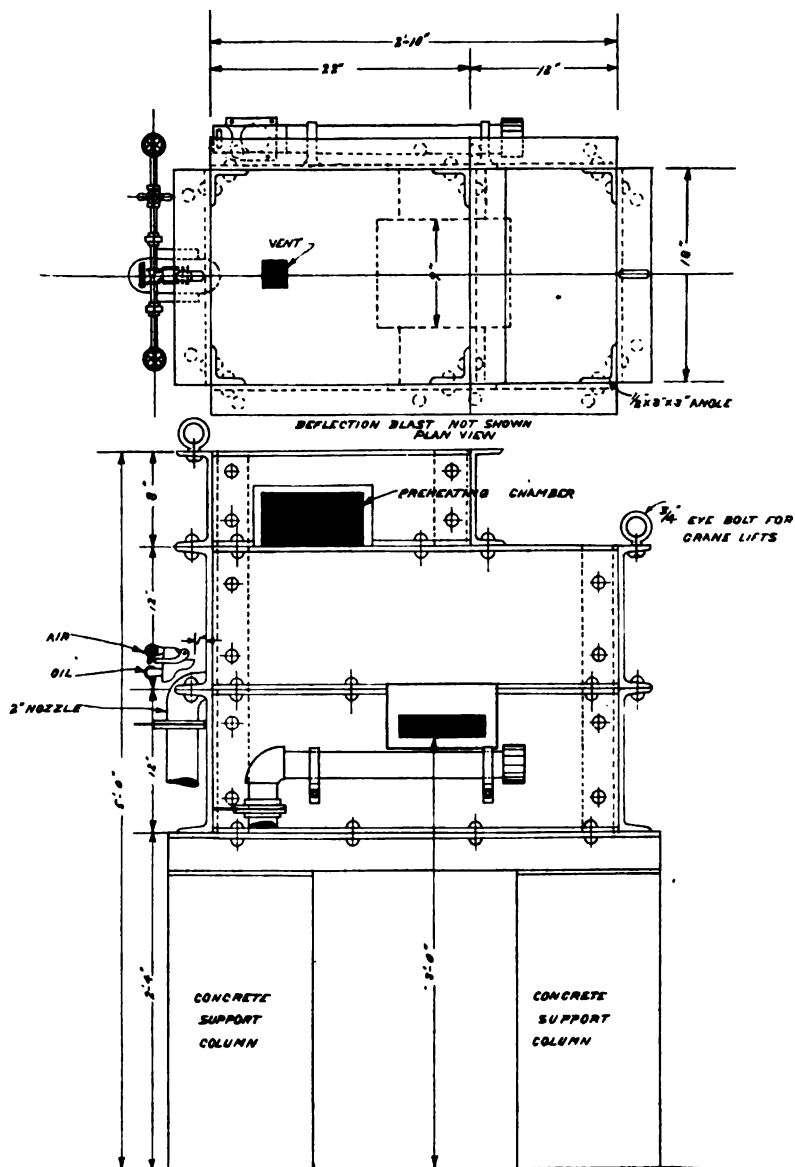


FIG. 4.—HIGH-SPEED TOOL-STEEL FURNACE.

Welding Furnaces

With coal- or coke-fired furnaces for this class of work, the pipe must be turned continuously; whereas, with an oil-fired furnace of proper construction, a flange can be welded on a 20-in. pipe without turning it; for the flame from the oil can be directed so that it will pass around the pipe and flange, producing a welding heat without moving the pipe. This furnace may also be used for vanstoning, etc. (See Fig. 5.)

Since it makes such a clean fire, oil is the best fuel for a billet-heating or scrap-welding furnace; and such a furnace formerly coal fired, when properly remodeled for the use of oil, will show a much greater output. Should there be any sulphur in the liquid fuel, it will not have any detri-



FIG. 5.—Welding Furnace.

mental effect upon the metal, if the combustion chamber is of the proper form and proportions.

Many people are under the impression that they can compare the relative values of coal and oil by calculating the calorific value of the two fuels. This, however, is not true. The only accurate way to test the two fuels is to compare them in various classes of service. For example, in welding safe ends on flues for locomotives or stationary boilers, with a modern flue-welding machine a blacksmith and two helpers can weld 16 flues per hour in a coal forge, whereas the same force of men can weld with an oil furnace 60 flues per hour. This is because three flues can be placed at a time in an oil furnace, and only one flue in a coal forge. More-

over, the smith must be constantly turning that one, to prevent the fire from burning a hole in it, and he must also put borax and sand upon the weld. The welds with modern oil-fired furnace are clean, it is not necessary to turn the flue, and it is unnecessary to place borax and sand upon the part being welded, the weld being perfect without it. In this class of service 58 gal. of oil are equivalent to 1 long ton of bituminous coal (2,240 lb.) with a calorific value of 14,200 B.t.u. per pound, while in boiler service the average is 147 gal. oil as equivalent to 1 ton of coal.

Foundry Practice

Some day not far in the future, oil will find its place in every gray-iron foundry in the United States. At present cupolas are used; but every one realizes that cast iron belongs to an unruly family and that it is materially effected by high or low temperatures. Again, the oxidation of the metal in coke-fired cupolas is excessive; but with an air furnace or other type of oil furnace, the oxidization is reduced to a minimum, the temperature of metal desired can be attained and maintained without variation from day to day regardless of climatic conditions, etc. I have prophesied that there is a great future for oil in this particular service, but, like every other new thought, it must be developed and perfected.

For ladle heating, oil as fuel in steel foundries, gray-iron foundries, malleable iron foundries, brass foundries, etc., is far superior to all other fuels. The various metals must be heated to certain temperatures before being poured and one of the new theories advanced in the year 1915 is that the ladles should be heated to approximately the temperature at which the metal is poured. This sounds reasonable; for if it is essential that the metal be at a certain temperature when poured, it is also equally important that it be not chilled when poured into the ladles. Oil is the fuel whereby the ladles can be properly heated to the same temperature as the molten metal.

Drying Ovens, Etc.

In mold-drying and core-drying ovens, liquid fuel is a desideratum, for with it we can maintain an even distribution of heat throughout the entire length of the oven, no matter what may be its proportions. Of course this is impossible with coal or coke. As the operator has the fire under perfect control, the material can be dried as quickly or as slowly as desired. Oil produces a more penetrating heat than other fuels, and if so desired, considerably less time may be used in this process without injury to the charge. Again, no stack is required, which reduces the expense both of construction and operation. In this practice, heat-recording instruments should always be used, as a matter of precaution as well as economy. The records should be in the office of the superintendent every

morning. It is surprising how much fuel can be saved by the use of heat-recording instruments and a sharp supervision of the work of the operator as indicated by the daily records.

The use of hot-air furnaces has had remarkable growth during the past two years, because they are clean and admirably adapted for the distribution of heat; moreover, they can be located in a small building some distance from the house, office, or factory, if desired, or they can be installed in the cellar or basement or in any other portion of the building. There is no fuel superior to oil in this class of service.

There has been a greater development in furnace construction during the past four years than in all the world's previous history. It requires 2,009 cu. ft. of air to furnish the oxygen requisite for the combustion of 1 gal. of oil. Only about 20 per cent. of this air is oxygen, while the other 80 per cent. is nitrogen and other gases which unfortunately must be heated up to the same temperature as the furnace, but, for economy's sake, must be expelled as quickly as possible. It is therefore necessary that the furnace construction be such that the consumed and inert gases will be expelled at once. The combustion chamber should be so located and of such form and proportions as to insure perfect reverberation of the heat, while aiding in the quick expulsion of the gases. Only one burner should be used on any billet-heating furnace, no matter whether the charging space be 12 by 18 in. or 8 by 24 ft. in size. For a regenerative melting furnace two burners are necessary; but of course only one burner is in operation at any one time. In all other melting furnaces, whether the bath be 2 ft. square or 20 by 140 ft., whether the product requires an oxidizing or a reducing flame continuously or alternately, only one burner should be used. Additional burners make the operation of the furnace too complex. While the operator may strive to run them all alike, some will be giving an oxidizing and others a reducing flame, thereby impairing the efficiency of the furnace.

In the designing of furnaces the question is not how many burners can be used, but how few will do the work. A poorly constructed furnace will ruin the efficiency of the best oil burner ever made, while a poor burner will ruin the efficiency of the finest furnace ever designed. It is as impossible to cover a flat furnace bottom or charging space with a round flame as it is to fill a square hole with a round plug. To obtain the highest efficiency and strictest economy it is necessary to have a good furnace, scientifically designed and constructed, as well as a superior type of oil burner.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Thermal Insulation of High-Temperature Equipment

BY P. A. BOECK,* NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

THE thermal insulation of high-temperature equipment for industrial purposes is a subject that has not received from engineers and designers the attention its importance deserves. This may be attributed to a number of causes: In the first place, heat flow is a rather difficult factor to measure under the working conditions of an industrial furnace, and until the comparatively recent introduction into certain operations of the heat balance sheet, with a systematic attempt to account for the discrepancies in the totals, the various causes of heat losses and the reasons for their existence were not brought to light. Moreover, such rapid advancement and radical changes in the design of equipment, especially in metallurgical lines, have occurred that practice has apparently outdistanced the theory of design. In the modern tendency toward greatly increased size of units, effort has seemingly been made to utilize the heat of fuel gases without proper attention having been paid to conserving heat energy and confining it strictly to the points of maximum activity.

The systematic study of the thermal properties of structural materials, which is being carried out by various government bureaus and also by a number of technical societies, is forming the basis for more intelligent and consistent work in the economically important subject of prevention of fuel waste.

Advantages of Insulation

It is not necessary in this practical and utilitarian age to enlarge upon the need of prevention of heat losses, but attention may advantageously be called to the other benefits derived simultaneously with the saving in fuel and the increase in thermal capacity of furnaces when properly insulated.

In most industrial furnaces—which are here spoken of in the broadest sense, indicating general high-temperature equipment for any purpose and heated in any manner—the operations are generally carried on through a temperature range which has been found by practice to give the most ef-

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fective results. If, in preventing the heat being dissipated from the outside of the furnace, the temperature throughout the furnace is made more uniform, the zone of reaction is manifestly greatly increased without additional fuel consumption. Furthermore, overheating at the source of heat, in the effort to bring as much of the furnace as possible into the zone of reaction, is materially reduced by preventing the loss of heat through the furnace walls. This has the ultimate effect of lowering the temperature at the source of heat and greatly increasing the life of the refractories—usually an important factor in high-temperature work. It is evident, too, that in furnaces which are surrounded by metallic casings or other equipment susceptible to the action of high temperatures, the deterioration and damage caused by overheating may readily amount to many times the cost of installing the proper protecting insulation. The humanitarian will also see the advantages gained by the protection of workmen from the unhealthy and unsanitary conditions brought about by being compelled to labor under overheated conditions.

The importance of thermal efficiency of all manner of high-temperature equipment is now being more generally recognized and is receiving much attention.

Heat Flow

Without going into a detailed consideration of the general laws governing the flow of heat, it would be well to call attention briefly to the following facts regarding heat flow, which will be illustrated by reference to the steam boiler as a well-known example:

The U. S. Bureau of Mines has treated¹ this subject in a remarkably clear and concise manner in Bulletin No. 18, to which reference should be had by those desiring to refresh their memories as to the principles of heat radiation, convection, and conduction.

The energy in the form of heat which eventually finds its way into the water in a boiler, for example, is, of course, subsequently available in other forms. That portion, however, which is transferred to the walls of the furnace setting by any of the three methods mentioned, is conducted through the setting, if means are not provided to prevent, and is lost by radiation or convection from the outer surface of the setting. It is the heat lost in this manner and methods for its prevention that will be especially considered in this discussion.

Rate of Heat Flow

The rate of transfer of heat under various conditions, by both conduction and radiation, shows the relative importance of these two methods in influencing heat losses. The amount of heat conducted through a unit

¹ Kreisinger, Henry, and Ray, Walter T.: The Transmission of Heat into Steam Boilers, *Bulletin No. 18, U. S. Bureau of Mines* (1912).

area from one part of the body to another is proportional to the difference in temperature of the two parts; directly proportional to the thermal conductivity of the body through which the heat passes and inversely proportional to the distance between the two parts of the body. In other words, the conduction of heat through a solid body from one plane to another is a direct function of the conductivity of the body and the difference in temperature of the two planes and an indirect function of their distance apart.² This is identical with Ohm's law for transfer of electrical energy.

The heat transferred from one body to another by radiation is proportional to the difference of the fourth powers of the absolute temperatures of the two bodies. While this is strictly true only of the ideal "black bodies," the variation is so small that for all practical purposes this relation holds good in ordinary procedure.³

Hollow Wall Construction

This relation indicates the reason why, in low-temperature work, such as refrigeration, etc., a hollow wall space is an effective insulator, whereas, in high-temperature operations, the loss of heat by radiation through a hollow wall space is so great that its insulating effect is less than if this wall space were filled with material of rather high thermal conductivity. This has been brought out by Ray and Kreisinger, who again demonstrate that the hollow wall space type of wall construction is much less effective as a means for preventing the loss of heat than a solid wall of any ordinary construction material of equal thickness. This is especially true if the air space in the hollow wall is near the furnace side and becomes highly heated. This is entirely contradictory to the general belief that, since air is a poor conductor of heat, air spaces built into the walls of a furnace will greatly reduce heat loss by radiation. While the heat does travel very slowly through the air by conduction, it leaps over the air space readily, by radiation, because the quantity of heat which passes across the hollow space is a function of the fourth power of the absolute temperatures of the surfaces inclosing it, which loss is enormously increased by rise in temperature.

Furnace Walls

In general, in high-temperature furnace construction, there are two separate and distinct factors which must be considered to produce an effective wall. The first of these is to provide a material having the ability to resist the action of high temperatures, sufficient mechanical

² Ray, Walter T., and Kreisinger, Henry: The Flow of Heat through Furnace Walls, *Bulletin No. 8, U. S. Bureau of Mines* (1912).

³ Gurney, Harold P.: Heat Radiation, *Journal of Industrial and Engineering Chemistry*, vol. iii, No. 11, p. 807 (Nov., 1911).

strength, and, possibly, the property of resisting corrosive slags, gases, etc., without spalling or being eroded. Secondly, to prevent the excessive loss of heat due to conduction from the interior of the furnace to the outside, where it is lost. It is rare that a good refractory material is a good insulator; usually it is necessary to augment or back up the refractory with some material having a much lower heat-conducting capacity. As a general rule, light porous substances are good thermal insulators; and, in a great majority of cases, this depends upon their entrapped voids, their apparent density being an excellent criterion of their conducting ability.

Requirements of Insulators

The requirements of the insulating backing for the more refractory lining are rather severe; an ideal insulator would have the following properties: It should be extremely high in insulating value, and sufficiently refractory so that no fusing or excessive shrinkage would take place in that portion which is in direct contact with the highly heated refractory wall. It should not be decomposed or change greatly in volume at that temperature, and it should, furthermore, be of light weight, unaffected by moisture, of convenient form, readily applied by unskilled labor, and low in cost. It should be of such composition as not to react upon or attack either the refractory material or the metal shell of the container, even in the presence of moisture; *i.e.*, it should contain no free acid radicals, and should not be broken or caused to settle by vibration or heat. It should not have high expansion and it should be sufficiently elastic to take up strains between the lining and the shell produced by temperature changes. While this is a rather formidable array of requirements, there are products upon the market which fulfill practically all of them.

Such materials as are being commonly applied to steam-temperature insulation, of which magnesia and asbestos are best known, are, of course, eliminated from consideration in the great majority of cases mentioned here. The lack of more extended practice in high-temperature insulation must be attributed to the fact that effective high-temperature nonconductors have not been generally available.

Amount of Insulation

When the kind of refractory material that is best suited to the furnace has been determined, the next most important item is the thickness of the walls and the nature of the insulating material most suitable, and it is necessary to determine the degree of insulation which will produce the most effective results. Hering has pointed out that, in order to effect a perfect insulation, the furnace should be surrounded with an insulating material that would maintain the same temperature at any point as that

of any adjacent point of the lining; in other words, that there should be a uniform amount of heat loss throughout the lining.⁴

In attempting to obtain perfect insulation, it is entirely possible to over-insulate, causing serious damage to the refractories in the high-temperature zones of the furnace. To maintain the inner walls, it may be necessary to permit a considerable flow of heat through the wall, with a corresponding decrease in the temperature of the refractories at the heated point, in order to prevent their destruction. It would be manifestly impracticable to insulate the roof of an electric steel furnace or open-hearth furnace, for instance, with a heavy layer of insulating material, because the cost of the refractory lining which would be destroyed would be considerably greater than the cost of the heat which would otherwise be lost.⁵

In electric-furnace practice, where extremely high temperatures are encountered and where effective insulation is necessary because of the relatively high cost of current, the question of insulation, or rather over-insulation, has been given more attention than in other lines. F. T. Snyder⁶ has presented a detailed study of the nature and amount of insulation which is permissible under various conditions and presents many valuable data which might with advantage be applied to other industries.

The necessity for effective insulation in stoves and other metallurgical equipment has been brought out strongly by Walther Mathesius, before this Institute, in a paper on High Blast Heats in Mesaba Practice.⁷

In discussing the thermal efficiency of the modern hot-blast stove and probable future methods of improvement, A. E. Maccoun, in a recent paper⁸ before the American Iron and Steel Institute, points out the advantages of effective insulation and the necessity for thermal insulation to increase the efficiency of the stoves.

The requirements of an ideal insulator have been outlined, and, in order to show the various methods of application which have been worked out (some of which have been in use abroad or in this country for some time), it will be well to select the insulator which apparently most nearly meets the requirements and use this as an example of methods of construction. In the applications shown, attempt was made to cover as wide a field of use as possible, to show the scope of possible applications.

⁴ Hering, C.: Thermal Insulation of Furnace Walls, *Metallurgical and Chemical Engineering*, vol. x, No. 2, p. 97 (Feb., 1912).

⁵ Lyon, D. A., Keeney, R. M., and Cullen, J. F.: The Electric Furnace in Metallurgical Work, *Bulletin No. 77, U. S. Bureau of Mines* (1914).

⁶ The Flow of Heat through Furnace Walls, *Transactions of the American Electrochemical Society*, vol. xviii, p. 235 (1910).

⁷ *Bulletin No. 99, Mar., 1915, pp. 539 to 555.*

⁸ Blast Furnace Advancement, *Transactions of the American Iron and Steel Institute* (1915).

Insulation

The insulator used, known as celite, on account of its extremely cellular nature, is a mineral product of a highly siliceous composition and of very light weight, which occurs on the Pacific Coast^{*} in an exceptionally pure state. It is composed of numerous hollow cells, and weighs, in its natural rock form, air dried, from 25 to 30 lb. per cubic foot. When this material is ground properly, so as not to destroy its cell structure, Sil-O-Cel powder is produced, which weighs but 8 lb. to the cubic foot and has a thermal insulating power about equal to that of cork, or from 10 to 12 times the insulating power of ordinary firebrick. In other words, a 1-in. layer of this material is the equivalent in insulating value of from 10 to 12 in. of firebrick. Being almost pure silica, its melting point is high, 2,930° F. (1,610° C.), as reported by the Bureau of Standards, and it can be subjected to high temperatures without fear of alteration. It has been found advisable, however, not to use celite as a refractory at extremely high temperatures without some direct protection. This is readily accomplished by using it as a backing material for more refractory and highly conducting bodies. Owing to its remarkable nonconducting properties, the accumulation of heat on its face is so great, owing to the fact that the surface is not cooled by conduction, that a "flash" of flame or gases might easily exceed the melting point of silica and cause failure. If it is protected, however, only modified and uniform temperatures are encountered, which are maintained without risk or damage.

It is possible further to prepare bricks and blocks of various sizes and shapes by sawing the natural material by means of gang saws. Standard 9-in. straight Sil-O-Cel brick made from natural celite weigh from 1½ to 2 lb. each and are equivalent in insulating value to many times their thickness of ordinary firebrick. In crushing strength, these brick withstand over 400 lb. per square inch and are sufficiently strong to stand transportation and handling.

The high insulating value of Sil-O-Cel bricks can be shown by applying the heat of a blast lamp or torch upon their surface for hours, the unheated surface remaining cool enough to permit of handling. Powdered Sil-O-Cel can be tested in the same way by packing it into a shallow box of convenient size.

The cost of these insulating bricks is but little more than that of firebrick, and of the powder about one-third as much, so that the first cost of this insulation is comparatively low. In fact, instances are on record where the entire cost of insulation has been saved in fuel in the first few weeks of operation.

^{*} Deposits worked by the Kieselguhr Co. of America, New York, Chicago, San Francisco, and Los Angeles.

Typical Wall Insulation

Because of the variation in form in which Sil-O-Cel products are supplied—that of brick, blocks of various shapes, in powdered form and as a plastic cement—this material is adaptable to almost any form of thermal insulation, as will be shown in the following typical examples:

General Types

In general, there are four forms of construction for high-temperature insulation which can be adapted to almost any character of equipment.

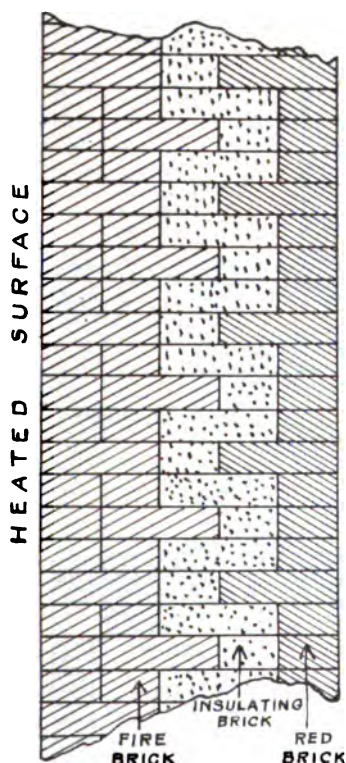


FIG. 1.—TYPICAL WALL CONSTRUCTION, SHOWING INSULATING BRICK LAID IN WALL.

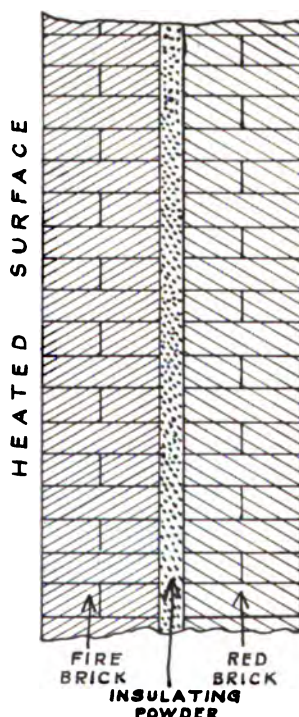


FIG. 2.—TYPICAL WALL CONSTRUCTION, SHOWING INSULATING POWDER IN HOLLOW WALL.

Fig. 1 indicates the usual method of using Sil-O-Cel brick interlaid between a course of firebrick and red brick for the prevention of heat leakage through walls. This form of construction is largely used in boiler settings, bakers' ovens, reverberatory-furnace walls and roofs, etc., and is generally applicable where a strong, solid, nonconducting wall is desired.

Fig. 2 indicates one of the methods of construction of an insulating

wall in which an otherwise hollow space is filled with insulating powder. From 2 to 4 in. are usually sufficient. The powder is packed slightly to a

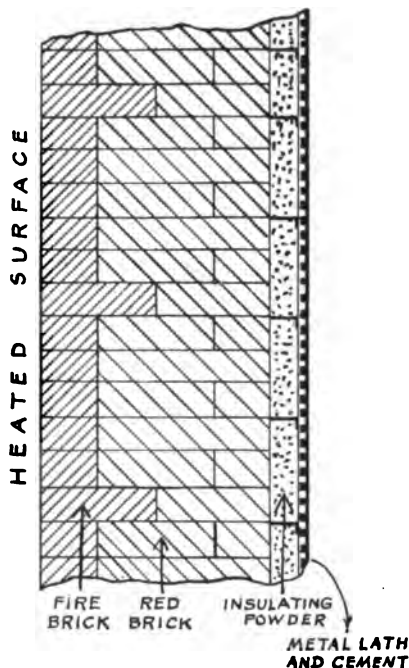


FIG. 3.—TYPICAL WALL CONSTRUCTION, SHOWING INSULATING POWDER SUPPORTED BY METAL LATH, CEMENT FINISH.

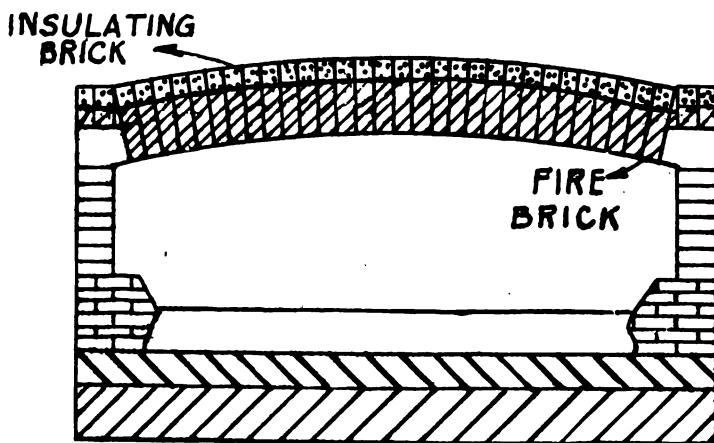


FIG. 4.—REVERBERATORY FURNACE ROOF COVERED WITH INSULATING BRICK.

density of approximately 12 lb. to the cubic foot, at which point it attains its maximum insulating value and is not subject to settling or contraction

due to either vibration or heat. Where this form of construction has been in severe service in high-temperature furnaces for a period of years no contraction or settling has taken place.

Fig. 3 indicates the method of insulating brick walls which are already in place. This form of insulation can be applied to old construction

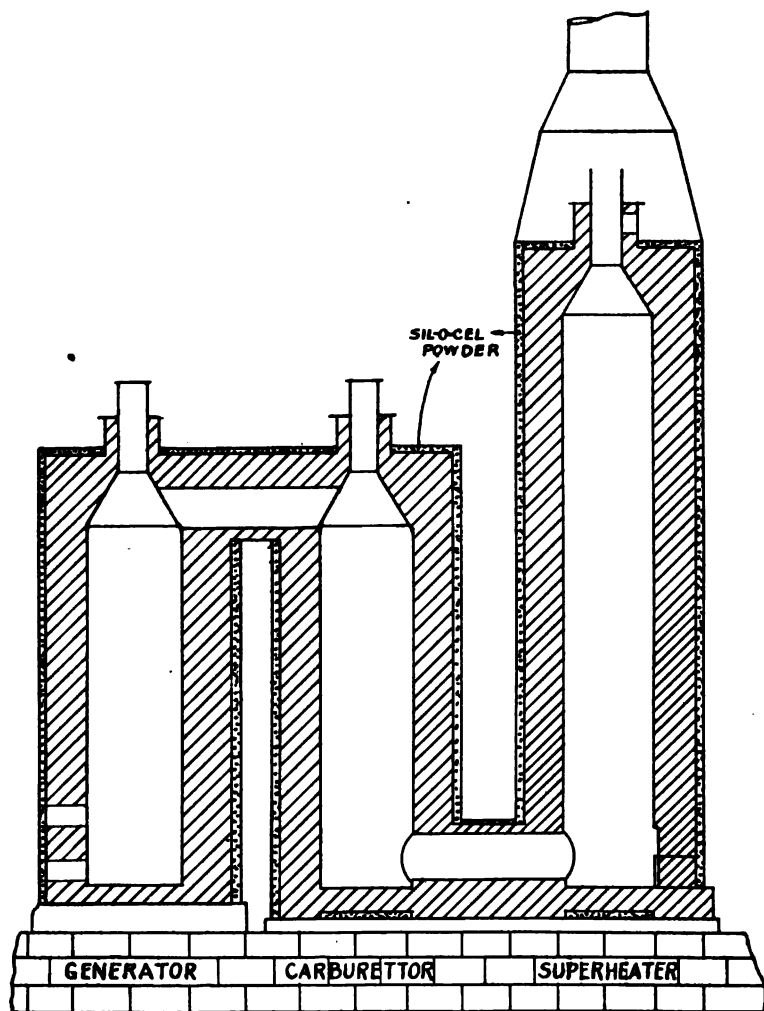


FIG. 5.—GAS-GENERATOR SET INSULATED BY MEANS OF POWDERED SILO-CEL.

as well as new. In this method, expanded metal lath is erected on angle irons at the required distance from the outer wall and coated on the outside with one or more coats of portland cement plaster, to which a small amount of Sil-O-Cel powder, approximately 20 per cent. by volume, has been added to give greater plasticity and ease of working and to in-

crease the heat-resisting properties of the cement. Sil-O-Cel powder is packed to a density of 12 lb. per cubic foot between the brick wall and the expanded metal lath. This form of construction is relatively inexpensive and allows of as much insulation as is required, and furthermore gives an absolutely permanent surface of excellent appearance, which can be applied to almost any character of equipment.

A layer of a pulverized material between the brick lining and furnace shell is the form of insulation most generally used in high-temperature

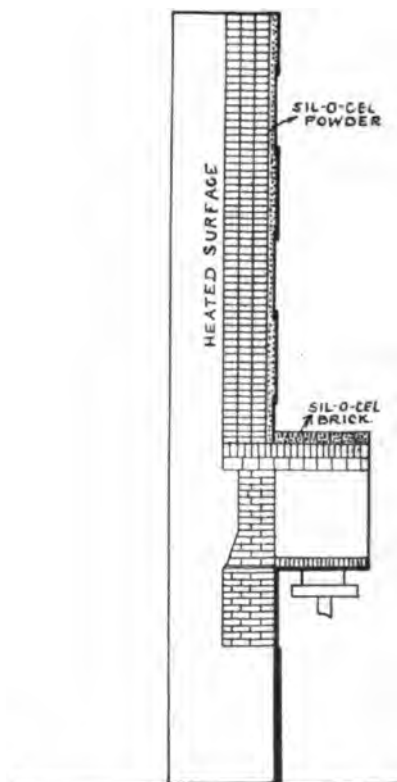


FIG. 6.—LIME KILN INSULATED WITH SIL-O-CEL POWDER.

metallurgical equipment, such as kilns of all kinds, gas generators, producers, stoves, etc.; in fact, it is universally applicable where the furnace has a suitable shell. In cases where support is required from the shell or setting, a denser variety of Sil-O-Cel, which is not compressible, has been found to be effective. The denser product does not possess the unusually high insulating values of the natural Sil-O-Cel, but offers mechanical advantages for certain construction.

These four forms of construction can be modified to give walls of any desired thermal efficiency, the respective methods being largely deter-

mined by the character of the work under consideration. Specific forms of high-temperature equipment are shown in Figs. 4 to 8.

In reverberatory furnaces Sil-O-Cel has found application as an insulating material for roofs and furnace walls in the manner indicated in Fig. 4.

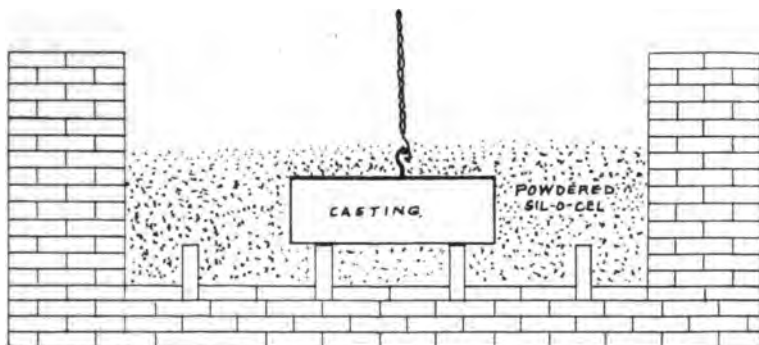


FIG. 7.—ANNEALING PIT CONTAINING POWDERED SIL-O-CEL FOR SLOW COOLING OF CASTINGS.

Fig. 5 indicates the method which is finding universal application for gas-making equipment, showing the powdered Sil-O-Cel tamped between the firebrick lining and the metal shell to a density of about 12 lb. to the cubic foot. This form of construction has been in actual use for a num-

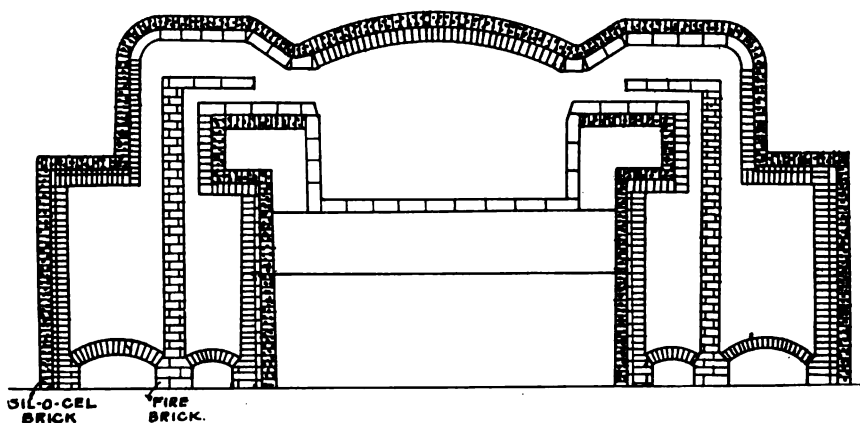


FIG. 8.—GLASS FURNACE INSULATED WITH SIL-O-CEL BRICK.

ber of years in some of the largest gas plants on the Pacific Coast, in insulating oil-gas generators, superheaters, carburetors and similar equipment. The advantages of Sil-O-Cel in this connection are threefold: In the first place, the heat conducted to the surface of the metal shell which is dissipated by radiation and convection is reduced to a minimum.

Secondly, in preventing the access of high temperatures to the steel shell, it prevents oxidation and deterioration of the shell through overheating and gives it the protection which it should have. Thirdly, Sil-O-Cel in the powdered form is to a certain extent elastic, allowing variation in expansion and contraction between the firebrick and the shell, eliminating all danger of excessive mechanical strains being set up between the two.

The insulation of lime kilns and similar equipment is shown in Fig. 6.

A unique application of the use of powdered Sil-O-Cel in annealing castings and other heat-treated metal forms is illustrated in Fig. 7, which shows an annealing pit partly filled with Sil-O-Cel powder, in which the castings are placed or suspended by chains until they are cooled to the proper degree for working. The annealing pit is built of brick and the depth of powdered Sil-O-Cel which is used is determined by the size and shape of the castings to be annealed and the rate at which cooling is desired. This form of annealing eliminates to a very large extent the elaborate and costly annealing furnaces which are used in a great many plants and bids fair to become one of the most important uses of powdered Sil-O-Cel. This material has also been used as a packing material in boxes in which the metals to be heat treated are placed, the entire box being heated and allowed to cool slowly.

Glass furnaces insulated as shown in Fig. 8 are considerably more effective and economical than those operating without insulation.

Before leaving this subject, it would be well to again mention the fact that no attempt has been made in this presentation to cover all the fields of high-temperature work, but merely to outline the methods which are applicable to almost any form of high-temperature equipment.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Hydro-Electrolytic Treatment of Copper Ores*

BY ROBERT RHEA GOODRICH, TUCSON, ARIZ.

(San Francisco Meeting, September, 1915)

EXPERIMENTS ON A PORPHYRY COPPER ORE FROM BISBEE, ARIZ.

THIS research was done partly in the non-ferrous laboratory of the Department of Metallurgy of Columbia University, under the direction of Dr. Edward F. Kern, and completed elsewhere. Acknowledgment is due to Prof. Arthur L. Walker, Dr. Edward F. Kern and Dr. William Campbell of the Department of Metallurgy, for their kind advice and for the inspiration derived from their instruction.

This report is given under the following heads: Petrographic Description (Microscopic Study); Sampling and Preparing Ore for Treatment; Chemical Analysis; Treatment of Ore.

PETROGRAPHIC DESCRIPTION

Texture: Variable. Fine to medium mixed aggregate.

Original structure: Obscure.

Secondary structure: Fractured; healed.

Mineralogy

(Minerals are grouped for interpretation purposes and are arranged in each group in approximate order of abundance)

Primary, Essential Minerals	Primary, Accessory Minerals	Secondary, Alteration Products
Obscure Probably quartz, and feldspar now wholly altered	Zircon	Quartz Kaolin Sericitc
Introduced Substances or Mineralization		Tertiary Changes and Enrichment Effects on Ores
Yellow sulphide (pyrite) Black sulphide (chalcocite) Quartz		Green malachite Limonite (a little from the yellow sulphide)

* Submitted in partial fulfillment of the requirements for the Degree of Doctor of Philosophy, in the Faculty of Pure Science, Columbia University.

The original character of the rock has been much obscured by modification. The variable texture indicates that the original rock was much brecciated. Silicification seems to have been a prominent feature. Some of the quartz appears to be remnants of primary grains. It must have been primarily an acid intrusive, which, after having been fractured and brecciated, has become still more acid by silicification. There appear to be recorded several sets of movements. The special features of most importance seem to be as follows:

The Ground Mass

This shows variable texture, fragmental (brecciated). The whole is so much modified that the fragmental character is not plain. Most of

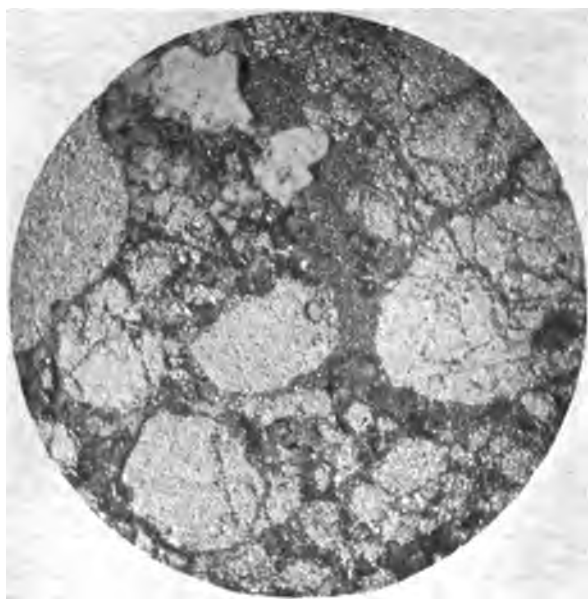


FIG. 1.—PHOTO-MICROGRAPH G₁.
Magnification 30 diameters.

the grains seem to be quartz. The only evidences of feldspar are those areas judged to be slightly kaolinized and areas of sericite. The fragmental character of the rock is somewhat further emphasized by the manner in which the metallics are distributed. These are distributed in such a way as to suggest an original fragmental (breccia), into the interstices of which were introduced the yellow and black sulphides. Certain streaked areas are much clearer than the rest of the ground mass and generally free from introduced metallics. These are filled with what is plainly

introduced quartz, and they seem to represent old fractures. It is difficult to say just what relation these bear to the mineralization periods; they may possibly represent the closing stages of silicification.

The Introduced Metallics

These consist of yellow pyrite and black chalcocite. The yellow sulphide seems to be a little too yellow for typical pyrite and not quite yellow enough for chalcopyrite. The yellow sulphide bears evidence of having been fractured and somewhat crushed. The fractures are healed with the black sulphide. The fractures extending across the grains of yellow sulphide end abruptly at their margins, and do not continue into



FIG. 2.—PHOTO-MICROGRAPH G₂.
Magnification 30 diameters.

the adjacent grains of ground mass. Under the microscope, there may be seen areas of yellow sulphide, veined and rimmed with black sulphide; the whole is rimmed with a narrow band of sericite and, in some cases, of quartz. It is evident that the history of this rock is very complicated.

It would seem, therefore, that two periods of mineralization are here represented: first, the decomposition of the yellow sulphide, followed by sufficient movement to cause fracturing and slight crushing, either during the closing stages of its deposition or immediately thereafter; second, the introduction of the black sulphide, which was deposited in the pre-

viously formed and fractured yellow sulphide and as an envelope surrounding the smaller grains of the yellow sulphide. Silicification very likely accompanied all of these changes. According to this interpretation, the rock is a silicified and mineralized brecciated, acid intrusive (quartz porphyry), in which two periods of mineralization are represented. There is no absolute proof in these slides as to the source of the secondary mineralization, *i.e.*, the chalcocite.¹

Photo-micrographs G₁, G₂, G₃XN, Figs. 1, 2, 3.

(Magnification 30 Diameters).

G₁—The large patches, light in shade, comprising the greater part of

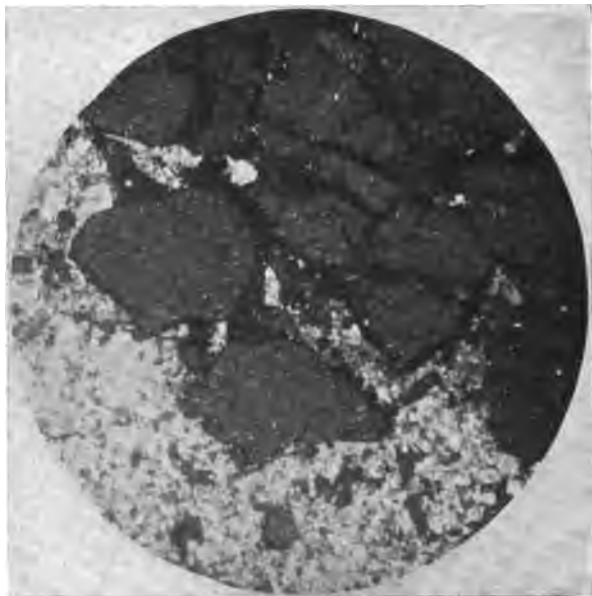


FIG. 3.—PHOTO-MICROGRAPH G₃XN.
Magnification 30 diameters.

the photo-micrographs, are yellow pyrite. There are two large patches, and several smaller ones, light in shade, but differing in appearance from those representing pyrite, in that they have a surface of homogeneous shade. These are quartz. The darker portions, occurring as spots and veinlets cutting the yellow pyrite, are of chalcocite.

G₂—In this photo-micrograph, the black masses and veinlets of chalcocite are very strongly marked. There is also some white non-metallic material.

G₃XN—This photo-micrograph was taken with crossed nicols of a

¹ This abstract was taken from the petrographic description by R. J. Colony, made under the direction of Dr. Charles P. Berkey, Columbia University.

slide containing more non-metallic material. The yellow pyrite here shows as large patches, quite dark, intersected by black veinlets of chalcocite. There is much white non-metallic material which, being doubly refracting, shows white as in the other photo-micrograph (taken with upper nicols out), while the pyrite, not being doubly refracting, shows darker.

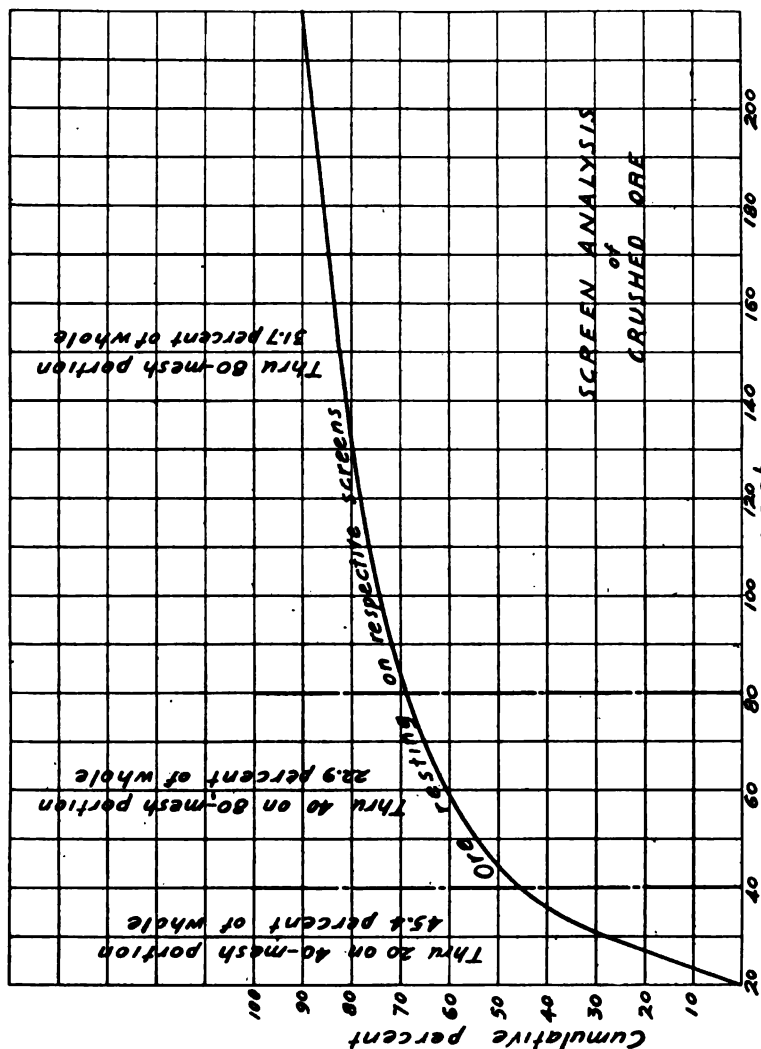


FIG. 4.—SCREEN ANALYSIS OF CRUSHED ORE.

SAMPLING AND PREPARING ORE FOR TREATMENT

The ore was received in lump form. Five hundred pounds were crushed to pass a 4-mesh screen, using the gyratory crusher followed by the cone-and-ring sample grinder. A sample for analysis was cut out

and ground through 100 mesh. One-quarter of the lot was then cut out for treatment. This was passed repeatedly through the sample grinder (which was set up so as to grind finer), until all passed a 20-mesh screen. This one-quarter was then thoroughly mixed by repeated coning, and then split by split shovel. One resulting one-eighth was retained for treatment and will be called "Whole through 20 mesh." The other resulting one-eighth was sized, producing the portions: Through 20 on 40 mesh, 45.4 per cent. of whole; through 40 on 80 mesh, 22.9 per cent. of whole; through 80 mesh, 31.7 per cent. of whole. (The screen analysis of the crushed ore is given in Fig. 4.)

Chemical Analysis

	Per Cent.	
SiO ₂ (insoluble).....	65.10	
Fe.....	10.90	
CaO.....	0.90	
Al ₂ O ₃	0.28	
Total Cu.....	6.04	$\left\{ \begin{array}{l} 1.70 \text{ sol. in dil. HCl.} \\ 4.34 \text{ bal. of Cu.} \end{array} \right.$
S.....	12.70	

TREATMENT OF ORE

It has been seen that the ore is mainly a sulphide. Copper sulphide ores, which contain an excess of silica, have been heretofore usually treated by mechanical concentration followed by smelting. But this ore, like much of the disseminated orebodies of the Southwest, contains a portion of its valuable copper contents in an oxidized condition—malachite. Mechanical concentration on pure sulphides makes usually a saving of 66 per cent.; and when there is an oxidized component the saving is likely to be still lower. For this reason, it was decided to make a test on this ore by a leaching method. The method selected comprises:

1. Oxidizing roast;
2. Leaching with dilute sulphuric acid;
3. Electrolytic precipitation of the dissolved copper.

Oxidizing Roast

A series of four roasts was made in a gas muffle furnace (17½ by 11½ by 4 in.—inside measurement). In order that the relative roasting qualities of coarse and fine material, as well as the most suitable temperature, might be determined, 300 g. of through 20 on 40 mesh material and 300 g. of through 80 mesh material were roasted, in two 7-in. roasting dishes. The temperature of roasting was measured by a thermo-electric pyrometer. Samples to determine the progress of roasting were taken at ½-hr. intervals. These samples were ground through 100 mesh and then tested as follows: One gram of sample was boiled in a covered casserole for 20

min. with 150 cc. of water. The soluble copper thus obtained is reported in per cent. copper soluble in water. After filtering and washing, the same portion of sample was boiled in a covered casserole for 20 min. with 150 cc. of dilute hydrochloric acid (100 cc. of concentrated hydrochloric

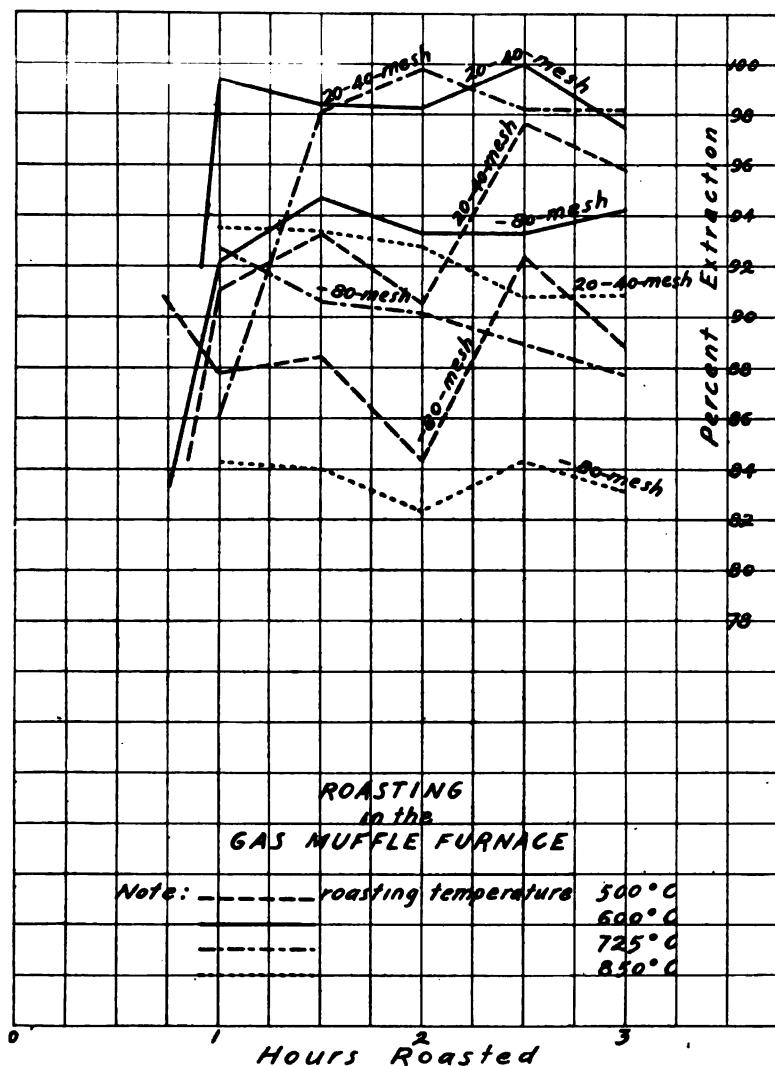


FIG. 5.—ROASTING IN THE GAS MUFFLE FURNACE.

acid diluted to 1,000 cc.). The soluble copper thus obtained is reported as per cent. copper soluble in dilute hydrochloric acid. The sum of these two is reported as the per cent. of total soluble copper. The difference between this and the total per cent. of copper found in the sample

is reported as per cent. insoluble copper. By dividing per cent. of total soluble copper by total per cent. of copper in sample, per cent. extraction was determined.

On referring to Fig. 5, Roasting in the Gas Muffle Furnace, where material through 20 on 40 mesh and material through 80 mesh were

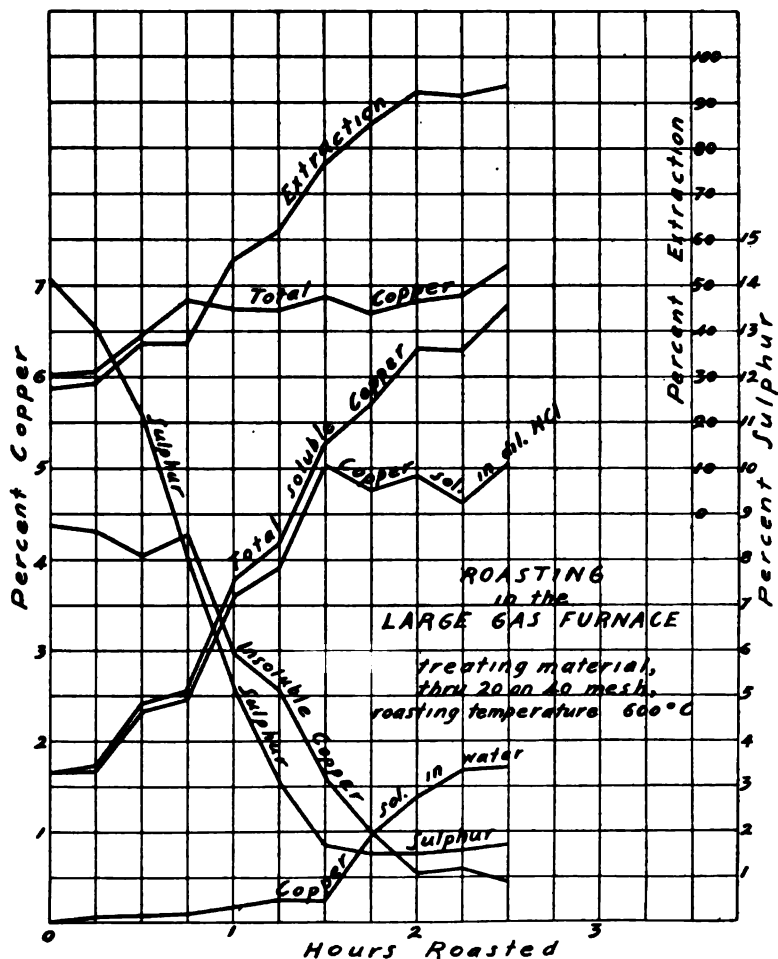


FIG. 6-A.—ROASTING IN THE LARGE GAS FURNACE.

roasted side by side in separate roasting dishes, it will be seen, by leaching with dilute acid, that the material through 20 on 40 mesh always gave a higher extraction than the material through 80 mesh. As regards temperature, the roast conducted at 850° C. gave the poorest extraction, the temperature being too high. (The series of roasts indicates that at temperatures above 725° C. the resulting copper oxide forms insoluble

compounds.) The roast conducted at 500° C. gave better extraction, but, as shown by subsequent roasts, the temperature was too low to secure the best extraction on leaching. Roasts conducted at 600° C. and 725° C. are equally good on material through 20 on 40 mesh; but on material through 80 mesh, the 600° temperature gave material which yielded the

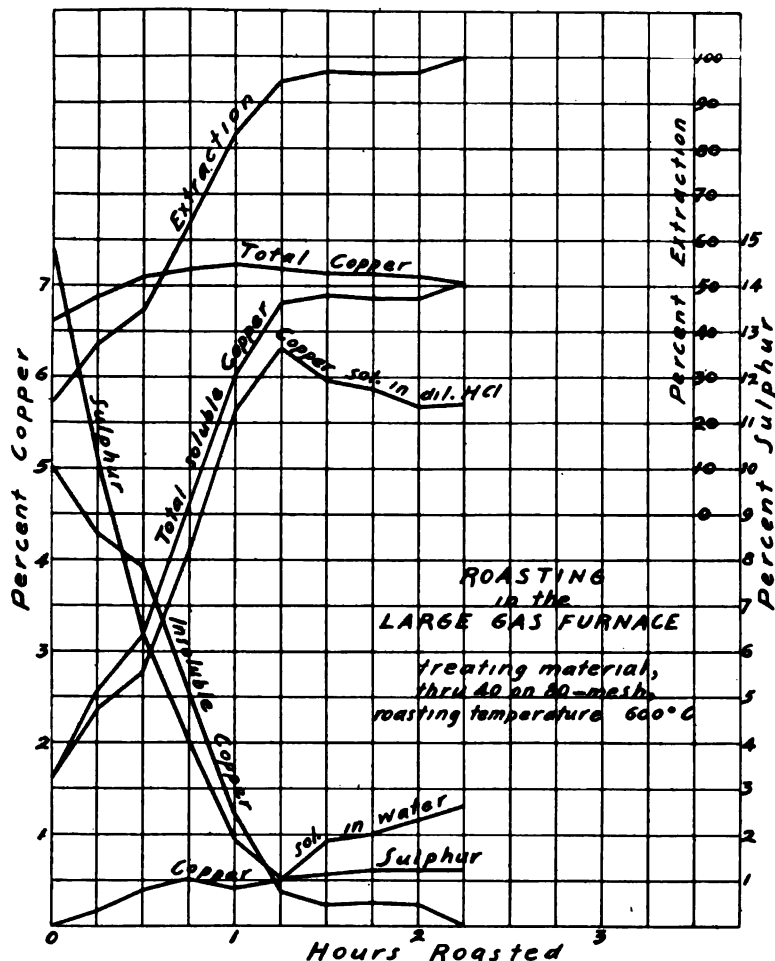


FIG. 6-B.—ROASTING IN THE LARGE GAS FURNACE.

best extraction. Consequently the temperature of 600° C. was selected as the best for subsequent roasting in the large gas roasting furnace.

There was no muffle in the large gas roasting furnace and its operation was similar to that of a gas-fired reverberatory furnace. The hearth, which was removable, consisted of a sheet-iron pan, lined with fire brick (22½ by 9 in.—inside measurement). Two series of roasts were made;

viz., A-Series and B-Series. Eight pounds of material were roasted per charge. The temperature of 600°C . was held constant. The materials roasted were: through 20 on 40 mesh; through 40 on 80 mesh; through 80 mesh; whole through 20 mesh. In all about 100 lb. of ore were roasted. Samples were taken during roasting at $\frac{1}{4}$ -hr. intervals. Of the several

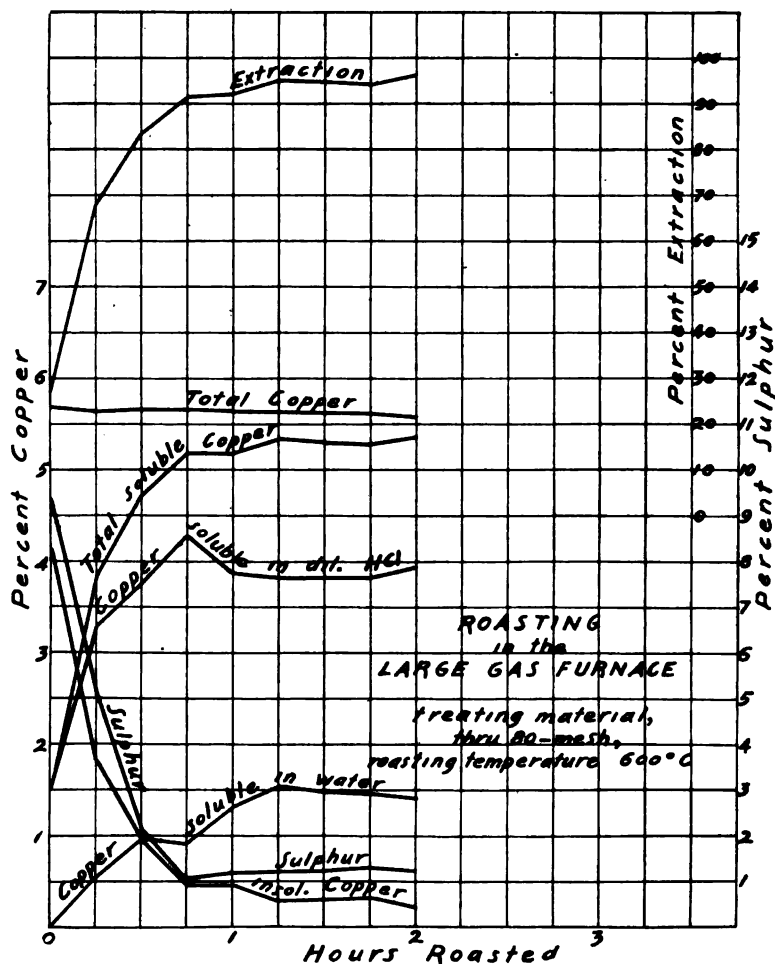


FIG. 6-C.—ROASTING IN THE LARGE GAS FURNACE.

roasts made of one size material, in the A-Series and in the B-Series as through 20 on 40 mesh, average samples were made. These samples were ground through 100 mesh, and tested for soluble copper, in the same manner as the samples resulting from the roasting in the gas muffle furnace.

Referring to Figs. 6-A, 6-B, 6-C, 6-D, Roasting in the Large Gas

Furnace, it will be seen that an extraction of 96 per cent. was secured in all roasts, except on the through 20 on 40 mesh material, which was but 93 per cent. The roasts were all carried on for a longer time than was necessary. An inspection of the curves will enable one to determine the most desirable time of drawing the roasts.

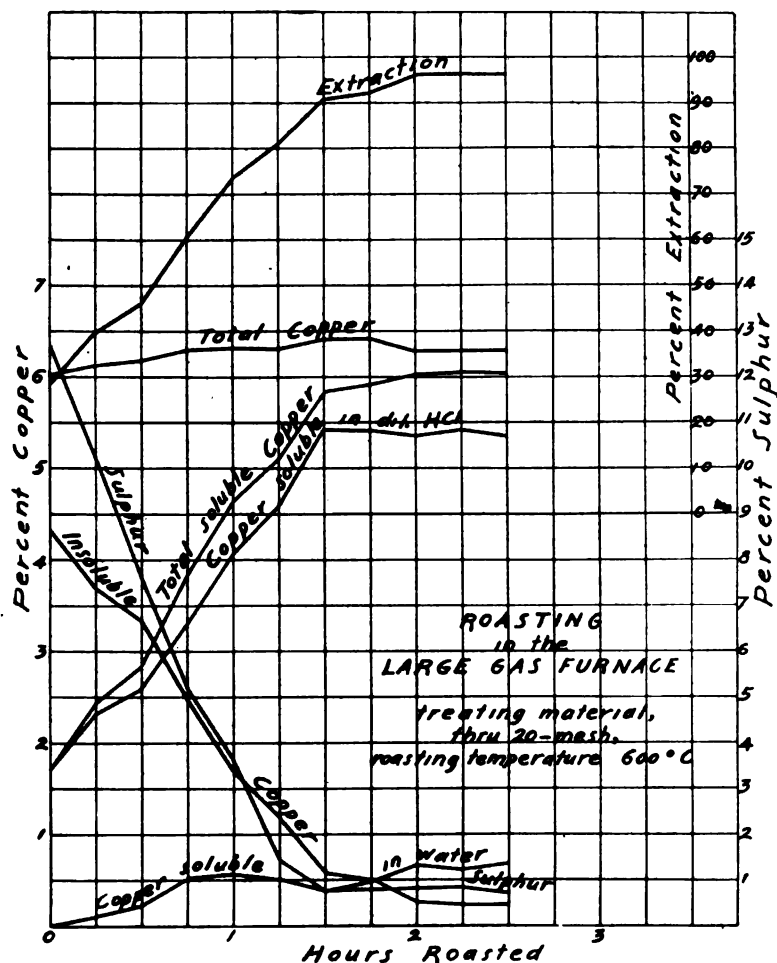


FIG. 6-D.—ROASTING IN THE LARGE GAS FURNACE.

Leaching with Dilute Sulphuric Acid

In order to forecast, as well as could be done in a small way, what might be expected in practice, regarding extraction and acid consumption, the following leaching experiments were made: 20 g. of roasted ore, to-

gether with 200 cc. of 10 per cent. H_2SO_4 ,² were introduced into glass-stoppered bottles, which were continually shaken. Two tests were made, one at a temperature of 21° C., the other at a temperature of 100° C. The material here leached was in the same condition, as regards degree

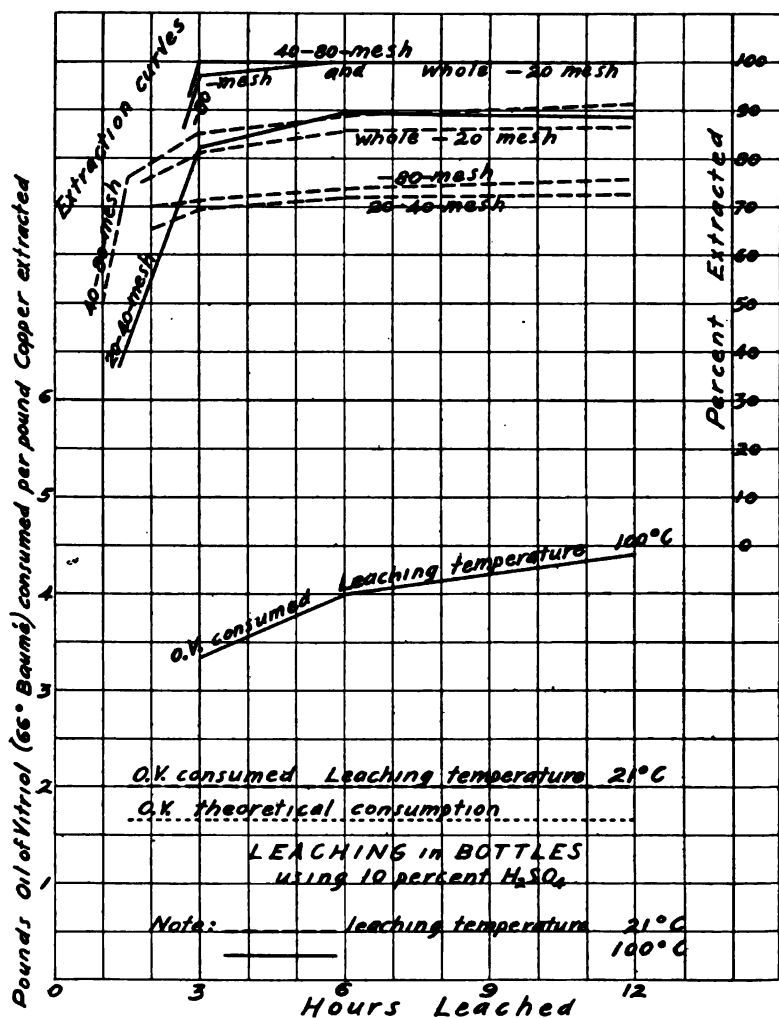


FIG. 7.—LEACHING IN BOTTLES.

of comminution, as when it left the sizing sieve, except as the roasting may have affected it. It is to be noted that it was not so with the samples

² In this paper per cent. of a constituent of a solution signifies grams of the constituent per 100 cc. of the solution: "per cent. H_2SO_4 ," signifies grams of free H_2SO_4 , per 100 cc.; "per cent. copper," grams of copper per 100 cc.

of roasted materials earlier tested for solubility of the copper contents, and which were all ground through 100 mesh before boiling in the casse-
role with water and dilute hydrochloric acid.

In the test made at 21° C., the extraction ranged from 72 to 90 per cent. The extraction was not improved by continuing the operation longer than 6 hr. The acid consumption was moderate, being 2.0 lb. of oil of vitriol (66° Bé.) per pound of copper extracted, as compared with 1.65 lb. theoretically required to dissolve 1 lb. of copper existing as oxide. The test which was made at 100° C. gave an extraction of 90 per cent. with the through 20 on 40 mesh material, while with all other materials the extraction was 100 per cent. The acid consumption was somewhat higher, as Fig. 7 shows. The time required for efficient leaching was between 3 and 6 hr. Nothing was gained by increasing the time of leaching beyond 6 hr.

The Electrolytic Precipitation

The Electrolytic Plant.—The electrolytic plant, designed for experimenting in the deposition of copper from solution with insoluble anodes, comprised a motor-generator set, circulating pumps, electrolytic cells, and a testing table. (See Figs. 8, 9 and 10.)

The motor-generator set used was a Robbins & Meyers $\frac{1}{4}$ -hp., 110-volt, 60-cycle, 1,750 rev. per minute, single-phase motor, belted to their 0.125-kw., 10-volt, D. C., 1,750 rev. per minute, compound-wound generator.

A small acid-proof pump could not be found in the market, so a single-acting acid-proof pump was designed. Four pumps were made. A 2-oz. syringe, No. 135, made by the American Hard Rubber Co., was used as a pump barrel. A block of dry maple was suitably bored. There were two ball valves; the seats were rings of $\frac{1}{4}$ -in. hard rubber; the balls, glass agates. The balls were ground into the seats with emery. The seats entered the cavities bored for them snugly. P. & B. acid-proof paint secured the seats in place so that there was no leakage. The syringe, the suction pipe, and the discharge pipe, entered holes in the maple block prepared for them. Water-tight connections were made by suitable soft-rubber packing rings. The Robbins & Meyers, $\frac{1}{8}$ -hp., 110-volt, 60-cycle, 1,750 rev. per minute, single-phase motor, through suitable pulleys and gears operated the four pumps, which made 35 strokes per minute. (See Fig. 10.)

The electrolytic cells used in the experiments of this paper were four cells with one cathode each, two cells with two cathodes each, and two cells with four cathodes each. Each cell had one more anode than cathodes. (See Fig. 9.) Figs. 11 and 12 show one cell with four cathodes, assembled and taken apart. The electrodes were spaced $\frac{3}{4}$ in. from center to center. Anode No. 6, seen in Fig. 13 (lower left-hand corner), was



FIG. 8.—TESTING TABLE.

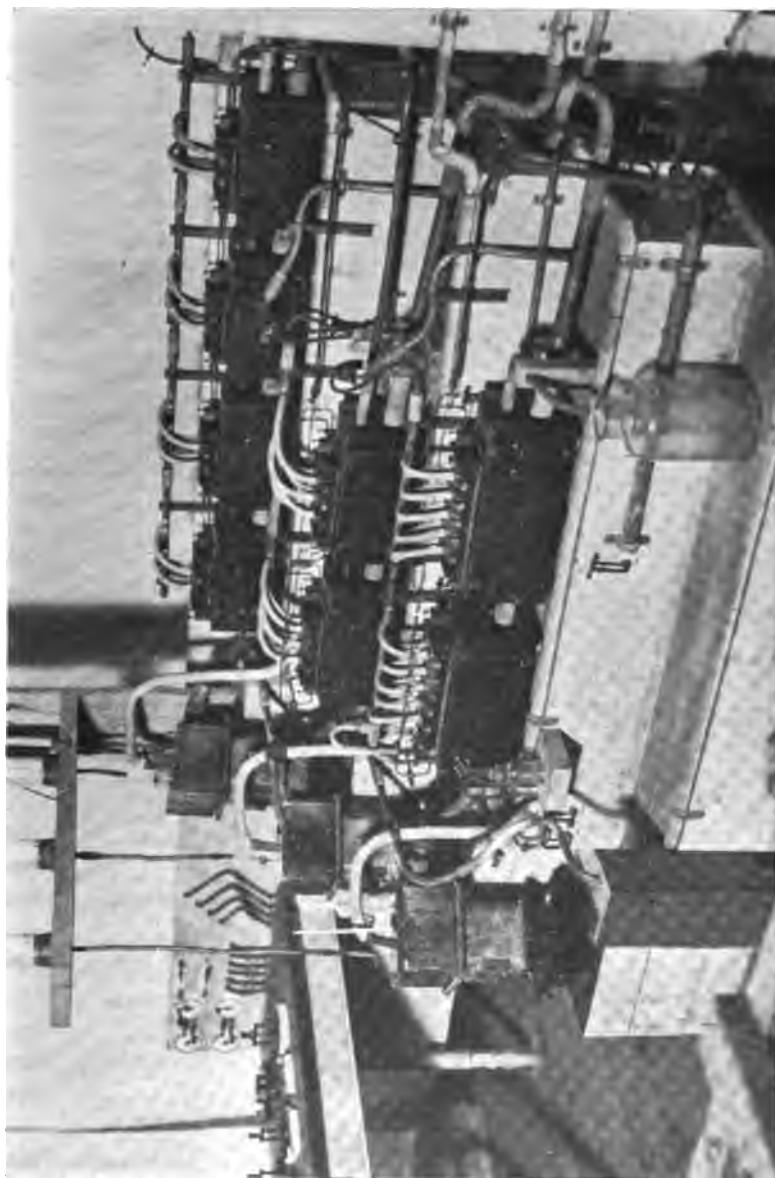


FIG. 9.—ELECTROLYTIC CELLS.

used in Tests Nos. 1 and 2. It was made from a sheet of lead (8 lb. to the square foot), $2\frac{1}{2}$ in. wide and of the proper length. The upper edge was wrapped around a piece of No. 6 bare copper wire by which it was suspended in the cell. Cathodes may likewise be seen in Fig. 13 (middle row). The cathode was made of a sheet of thin copper attached to a copper clamp by which it was suspended in the cell. There were four insulating blocks of hard rubber, 1 by 1 by $\frac{1}{2}$ in., beneath the cell. There were four pieces of hard rubber, 1 by 1 by $\frac{5}{16}$ in., with a $\frac{3}{16}$ -in. upward projection for insulating and holding the bus-bars in place. In Fig. 12, just below the cell, the bus-bars may be seen. The bus-bar was comprised of a 1 by 1 in. brass angle and a 1 by $\frac{1}{4}$ in. hard-rubber strip. The rubber strip was bolted to the vertical leg of the brass angle. There were two pieces of hard rubber spacing pieces used for holding the bus-bars rigidly in place. The method of connecting the electrodes with the bus-bars was similar to practice. The brass and the hard rubber of the bus-bars were so cut that when the horizontal arms of the electrodes rested on them, on one side the arms of the anodes rested on the brass plus conductor while the arms of the cathodes were insulated by hard rubber, and on the other side the arms of the anodes were insulated by hard rubber while the arms of the cathodes rested on the brass minus conductor. The binding posts with No. 6 bare copper wire served for connecting between cells. (See Figs. 9, 11, and 12.)

Quite elaborate piping systems about the cells may be seen in Fig. 9. The elevated temperature tank and the cells of a step were connected by 1-in. lead pipe, the connections to and from the step circulating pump were made by $\frac{1}{2}$ -in. lead pipe. For the progressive circulation of the electrolyte through the plant, $\frac{1}{2}$ -in. lead pipe was used. The sulphur dioxide gas was conveyed to the cells by its system of lead piping. The gas was supplied from a cylinder of liquid gas. The gas cylinder was connected by small rubber tubing to the end of a 2-ft. length of 1-in. lead pipe. From the rear side of this pipe, four $\frac{1}{2}$ -in. lead pipes, leading to different parts of the plant, passed beneath the cells. (See Fig. 9, in foreground.) There was placed horizontally above the cells of each step a $\frac{1}{2}$ -in. lead pipe. This pipe was fitted with $\frac{1}{4}$ -in. tee connections which were connected to the hollow anodes, used in Tests Nos. 3 and 4, by rubber tubing. Bottles were introduced between the ends of the four distributing pipes and these horizontal pipes, just mentioned, in order to indicate the amount of gas flowing. The four indicator bottles for judging the rate of flow of the gas contained a little water through which the gas bubbled. A screw clamp on the rubber tubing connection, between each distributing pipe and its indicator bottle, adjusted the distribution of gas.

A poplar kitchen table, with top 30 in. by 48 in., was selected for the testing table. A Weston miniature, precision, direct-current volt-am-

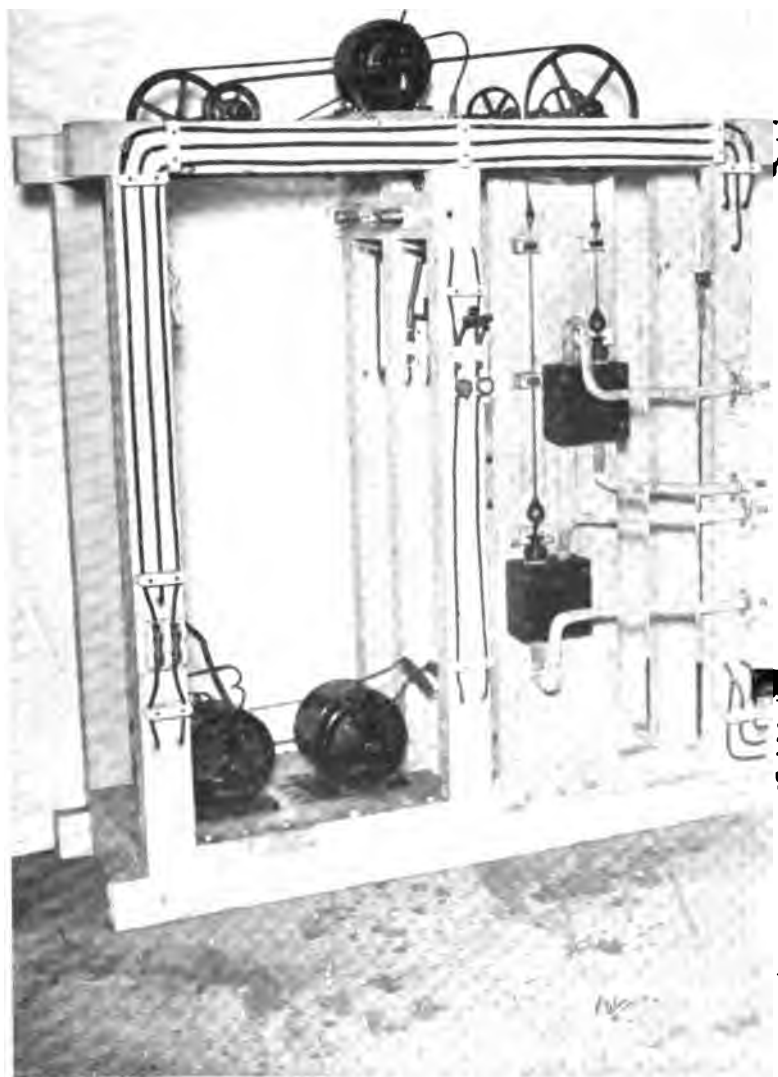


FIG. 10.—FRAME.

meter was used (Model 280 triple-range portable volt-ammeter, Weston Electrical Instrument Co., Newark, N. J., Bulletin No. 8, 1912). The scales were: 150, 15, 3 volts; 30, 15, 3 amperes. The instrument was permanently placed and connected on the table top at the operator's left hand. The table was wired and connected with the rest of the plant, so that all the readings desired might be taken on the one instrument when the switches were properly manipulated. The following readings could be taken:

1. Total amperes, load on generator.
2. Amperes, No. 1 load circuit. (Plant could be operated as one circuit or as two parallel circuits.)



FIG. 11.—CELL ASSEMBLED.

3. Amperes, No. 2 load circuit. (Plant could be operated as one circuit or as two parallel circuits.)
4. Generator volts.
5. Individual volts across cells.

There were fourteen voltage wires, each terminating at one end (the cell end) in a tee connector. In the foreground of Fig. 9 may be seen the cell end of one voltage wire (No. 12). In connecting up the cells (arrangements might be varied in different tests), there was always to be found close to the cell the cell end of a voltage wire. In making series connections between consecutive cells, No. 6 copper connecting wires from the respective binding posts of the bus-bars of the two cells entered a tee connector, one leg of which was permanently attached to the end (cell

end) of a voltage wire. All the fourteen voltage wires passed to the testing table where they connected to two 13-point circular switches. The switch to the operator's left had connected to it wires Nos. 1 to 13, while the switch to his right had connected to it wires Nos. 2 to 14. On properly setting the two 13-point switches and throwing the three-pole double-throw switch adjoining, the individual volts across cells, or the

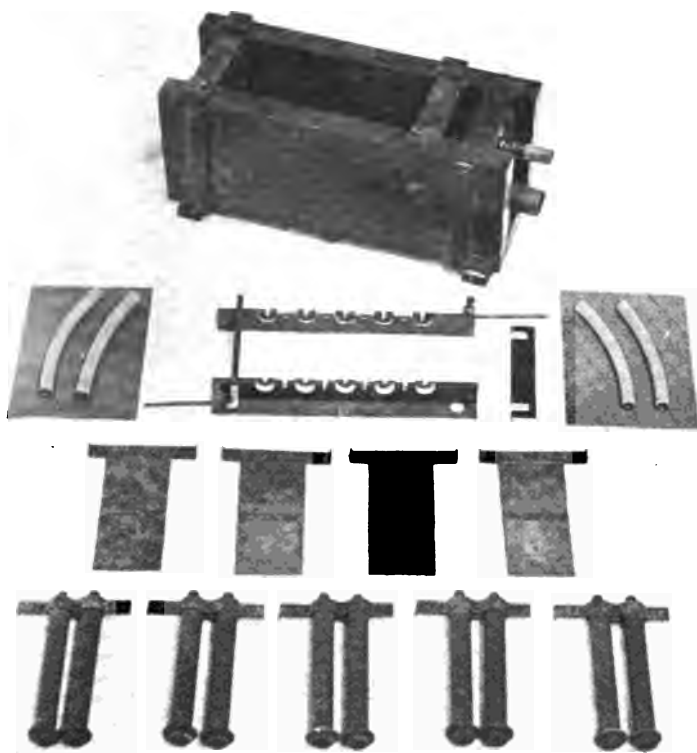


FIG. 12.—CELL TAKEN APART.

sum voltage of any number of cells, could be recorded by the instrument on suitable scale. Not all the cell ends of the voltage wires were connected in any one test. When the plant was wired up for some one test, the voltage wires in use were noted and the corresponding switch points on the table top were tagged for the operator's use. All switch points connected with unused voltage wires were dead.

The generator field rheostat is seen in the foreground of Fig. 8. To the operator's left, on the vertical panel, were four battery rheostats, con-

nected in parallel, serving as a variable load in series with the cells in load circuit No. 2. These battery rheostats could be short circuited, when the generator field rheostat would give the desired voltage regulation. When a lower voltage was desired than the field rheostat could give, then the variable resistance in series was used.

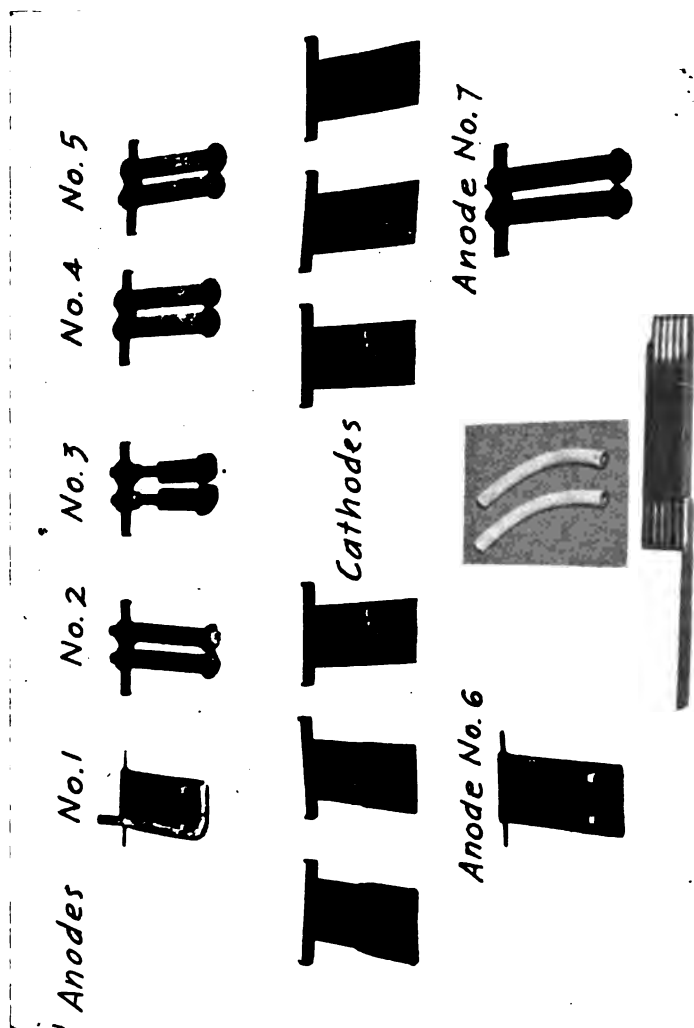


FIG. 13.—ELECTRODES.

Test No. 1.—Arrangement A, in "Example of Step Arrangement of Plant," was employed, operating No. 1 Step and No. 2 Step and omitting the Finishing Step. No. 1 Step had four cells, connected in series, each with one cathode. No. 2 Step had two cells, connected in parallel, each with two cathodes: this combination will be spoken of as one cell of four

TABLE I. TESTS Nos. 1, 2, 4.

[illegible]

Note: For electrical record of Finishing Step, Tests No. 2 and 4, see Fig. 15

cathodes. Note that the plant was operated with one-third as many cells as are indicated in the diagram, Fig. 16 (Example of Step Arrangement of Plant—A). Evaporation was compensated for by adding sufficient water to the temperature tank of No. 1 Step and to the temperature tank of No. 2 Step. The relative amounts of water for each step were proportional to the step surface exposed to evaporation. Consequently the overflow to the sump of No. 2 Step was equal in volume to that of the copper solution fed to the temperature tank of No. 1 Step. (See Table I.)

Current density of 20 amperes per square foot was selected for No. 1 Step. The area (sum of the two sides) of the cathode is 0.0824 sq. ft., therefore a current of 1.648 was supplied. The volume of copper feed solution required for 12 hr., in order to supply the cells of No. 1 Step with an amount of copper equal to that deposited in that step by the current selected plus the copper carried on by the progressive circulation so that the electrolyte would remain constant in composition, was determined as follows:

$$\frac{1.648 \times 4 \times 12 \times 1.1855 \times 0.90}{0.75 \times 0.06} = 1,872 \text{ cc. copper feed solution required for 12 hr.}$$

Where,

Current amperes.....	1.648
Number of cells in series, No. 1 Step.....	4
Duration, hours.....	12
Theoretical copper deposited by 1 ampere-hour, grams.....	1.1855
Current efficiency, per cent., assumed.....	90
Copper deposited in No. 1 Step, expressed as per cent. of that in feed.....	75
Copper in feed solution, per cent.....	6

1,872 cc. of No. 1 Step electrolyte, containing 1.5 per cent. copper, overflowed from the last cell of No. 1 Step to the temperature tank of No. 2 Step. This supplied No. 2 Step with an amount of copper equal to that deposited by the current in that step, plus the copper carried on to the sump by the progressive circulation, as the following calculation shows:

$$\frac{1.648 \times 1 \times 12 \times 1.1855 \times 0.90}{0.75 \times 0.015} = 1,872 \text{ cc., overflow of No. 1 Step electrolyte, required for No. 2 Step.}$$

Where,

Number of cells in series, No. 2 Step.....	1
Copper in overflow of No. 1 Step electrolyte, per cent.....	1.5

The strength of the electrolyte in H_2SO_4 , which is developed by the acid liberated by the copper on deposition, was determined as follows:

No. 1 Step.

$$0.06 \times 0.75 \times 1.54 = 0.069 \text{ (or 6.93 per cent.)}.$$

Where,

Copper in feed solution, per cent.....	6
Copper deposited in No. 1 Step, expressed as per cent. of that in feed.....	75
H ₂ SO ₄ freed per unit of copper deposited.....	1.54

No. 2 Step.

$$0.06 \times 0.9375 \times 1.54 = 0.0866 \text{ (or 8.66 per cent.)}.$$

Where,

Copper deposited in Nos. 1 and 2 Steps, expressed as per cent. of that in feed.....	93.75
--	-------

In starting up the plant, each step was supplied with the necessary volume of electrolyte to put it in operation. The composition of this initial charge of electrolyte in copper and acid was that which the above calculations indicate and which may be seen in Table I. The results of this test show that, when employing the Step Arrangement of Plant, and when adding water to each step to compensate for evaporation, the initial values of copper and acid contents of the electrolyte remain constant during the electrolysis. The results of this test also show the performance when copper is deposited, using insoluble lead anodes. (See Fig. 9; Tables I and II; and Fig. 16, Diagram Example of Step Arrangement of Plant—A.)

Test No. 2.—Arrangement A, in "Example of Step Arrangement of Plant," was employed, using No. 1 Step, No. 2 Step, and the Finishing Step. The arrangement of the cells in Nos. 1 and 2 Steps was the same as in Test No. 1. The Finishing Step had two cells, connected in parallel, each with four cathodes; this combination will be spoken of as one cell of eight cathodes. The operation of the Finishing Step of this test will be discussed in Test No. 4.

In order to simplify the operation, a weaker copper feed solution was made for No. 1 Step by adding 1,170 cc. water to 1,872 cc. of 6.0 per cent. copper solution, producing 3,042 cc. of feed solution containing 3.7 per cent. copper, which was used for the 12-hr. run. This supplied No. 1 Step with the same amount of copper as in the previous test and with the water necessary to compensate for evaporation. There should have overflowed from Step No. 1, during the 12-hr. run, 1,872 cc. of solution containing 1.5 per cent. of copper and 6.93 per cent. of H₂SO₄. When, as in Test No. 1, evaporation in No. 2 Step is compensated for by the addition of an equivalent amount of water, this becomes the step electrolyte with 0.375 per cent. of copper and 8.66 per cent. of H₂SO₄. But in this test no water was added to No. 2 Step, so assuming evaporation to be the

same in amount as during the preceding test, namely 925 cc., there should have overflowed to the sump $1,872 - 925$, or 947 cc. solution of the following composition:

$$0.00375 \times \frac{1872}{947} = 0.00743 \text{ (or 0.743 per cent.) copper.}$$

$$0.0866 \times \frac{1872}{947} = 0.171 \text{ (or 17.1 per cent.) } \text{H}_2\text{SO}_4.$$

Note that in Test No. 2, as well as in Test No. 4, the initial charge of electrolyte to all steps deviated slightly in H_2SO_4 composition from the value which was calculated that it should have been. There was charged to No. 1 Step, a solution containing 6.24 per cent. H_2SO_4 , instead of 6.93 per cent.; to No. 2 Step and to Finishing Step, 15.4 per cent. H_2SO_4 , instead of 17.1 per cent.

The results of this test show that, when employing the Step Arrangement of Plant as outlined above and which differs in some details from the method used in Test No. 1, and when supplying initial charges of electrolyte of the compositions stated, these initial values remain constant in both copper and acid. The results of this test also show the performance when copper is deposited, using insoluble lead anodes. (See Figs. 9, 15, and 16; Tables I and II.)

Test No. 3.—But one electrolytic cell was used in this test. The intention was to introduce sulphur dioxide gas into the electrolyte during electrolysis in all subsequent work, so this test was run as a preliminary test to determine the best method of introducing the gas as well as the most suitable anode, in order to secure the best depolarizing effect.

Kinds of Anodes.—See Fig. 13: Anodes Nos. 1, 2, 3, 4 and 5, top row, counting from left to right; Anodes No. 6, lower left corner; No. 7, lower right corner.

Anode No. 1 was the lead Anode No. 6, used in Tests Nos. 1 and 2, so modified that SO_2 could be introduced into the electrolyte. Sufficient width and length was cut from one side and the bottom of Anode No. 6, as formerly used, so as to permit of attaching a $\frac{1}{4}$ -in. lead pipe, without increasing its size. This lead pipe was closed at the immersed end, and had its horizontal leg perforated on the upper side with three No. 50 drill holes (about $\frac{1}{16}$ -in. diam.), equally spaced. It was thought that the placing of the perforations thus would give an even distribution of the gas over the surface of the anode. (See Fig. 13, Anode No. 1.)

Anode No. 3 was made of lead pipe to imitate in form the carbon anode (No. 7). The main part is a piece of a $\frac{1}{2}$ -in. lead pipe. The bottom end was closed with a disk of lead in which was drilled a $\frac{1}{8}$ -in. hole (the same size as the central channel in the carbon anode). A short piece of $\frac{1}{4}$ -in. lead pipe was forced into and burned to the upper end of the $\frac{1}{2}$ -in. lead pipe. To this $\frac{1}{4}$ -in. lead pipe rubber tubing was attached

for supplying the sulphur dioxide gas. A soft rubber washer was placed on the pipe, near its lower extremity, to act as an insulator for preventing the occurrence of short circuits in the cell. Such a rubber washer was similarly placed on all other round anodes. This soft rubber washer had the same office as the hard rubber pegs of the flat lead anode (No. 6). The complete anode comprises a pair of these lead pipes which are rigidly held in a special copper clamp. (See Fig. 13, Complete Anode No. 2.)

In Anode No. 3, a $\frac{1}{4}$ -in. lead pipe extends through a short section of a $\frac{1}{2}$ -in. lead pipe, which enables it to be held firmly in the standard copper clamp. A piece of charcoal free from flaws was cut exactly $\frac{3}{4}$ in. square by $2\frac{1}{2}$ in. long, and bored so as to fit snugly over the $\frac{1}{4}$ -in. lead pipe. The sulphur dioxide gas enters the electrolyte from the bottom of the $\frac{1}{4}$ -in. lead pipe. (See Fig. 13, Anode No. 3.)

Anode No. 4 is of carbon. The carbon tube from which the anode was made is $\frac{3}{4}$ in. outside diameter, $\frac{1}{8}$ in. inside diameter and 12 in. long. This is an electric-arc carbon, incomplete in manufacture, and made by the National Carbon Co. Were it completed, the center channel would be filled with another grade of material, and then it would be a standard carbon for use in an electric arc for industrial purpose. This anode had a number of small holes made with a No. 50 drill, entering radially to the center channel. (See Fig. 13, Anode No. 4.)

Anode No. 5 differs from Anode No. 4 in that saw cuts were made instead of drill holes. These saw cuts extend half way through the carbon tube intersecting the center channel. (See Fig. 13, Anode No. 5.)

Anode No. 7 differed only from Nos. 4 and 5 in that it had no lateral perforations, the gas entering the electrolyte from the bottom extremity of the center channel. (See Fig. 13, Anode No. 7.)

The Depolarizing Effect of Sulphur Dioxide Gas.—On the assumption that all chemical energy is transformed completely into electrical energy, it was found by calculation that when no depolarizer is used, in a copper sulphate electrolyte, the theoretical counter E. M. F. of polarization is 1.22 volts, and when sulphur dioxide is used as depolarizer, the theoretical counter E. M. F. of polarization is -0.15 volt.³ Thus theoretically, sulphur dioxide gas should reduce the polarization by 1.37 volts, changing the polarization of the cell from 1.22 volts opposing the generator to 0.15 volt, pulling with the generator. Consequently in this experiment a reduction of polarization of 1.37 volts has been taken as the standard of perfection. In the calculation of the counter E. M. F. of polarization, the correction term of absolute temperature times the temperature coefficient of the electrical energy was neglected, because its value is unknown. The results obtained were somewhat influenced by the omission of this correction term.

Polarization volts were determined as follows: During the test, when

³ *Transactions of the American Electrochemical Society*, vol. xxv, p. 230 (1914).

running at the current density desired, Reading 1 was taken of amperes and volts. By suitably manipulating the generator field rheostat, together with the series resistance in the circuit, the current was reduced appreciably. New current value, together with corresponding volts, was quickly recorded, giving Reading 2. The current was quickly brought back to its running value when Reading 3 was again taken of amperes and volts. Reading 3 should correspond to Reading 1. If it did not, the system was allowed to assume again its normal constant conditions and the same operation was repeated. All the readings were taken in a few seconds.

Example illustrating determination of polarization:

Reading 1, at C. D. 8 amperes per sq. ft.,

$$I_1 = 2.637 \text{ amperes; } V_1 = 0.710 \text{ volt.}$$

Reading 2,

$$I_2 = 1.000 \text{ amperes; } V_2 = 0.533 \text{ volt.}$$

$$I_1 - I_2 = 1.637 \text{ amperes; } V_1 - V_2 = 0.177 \text{ volt.}$$

Then

$$V_1 = E + I_1 R$$

$$V_2 = E + I_2 R$$

$$V_1 - V_2 = (I_1 - I_2) R$$

$$R = \frac{V_1 - V_2}{I_1 - I_2} = \frac{0.177}{1.637} = 0.108 \text{ ohm.}$$

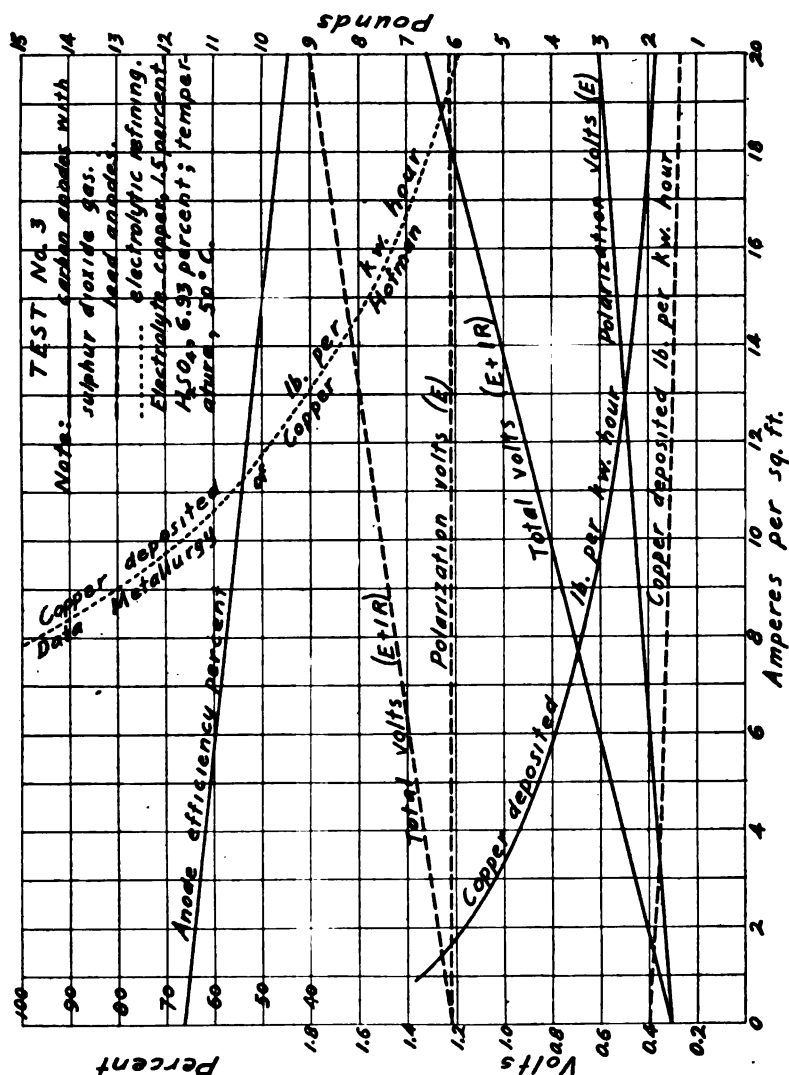
$$E = V_1 - I_1 R = 0.710 - (2.637 \times 0.108) = 0.425 \text{ volt.}$$

R equals ohmic resistance, made up of resistance in the solid conductor, contact resistance, and resistance in the electrolyte. This is constant.

E equals polarization voltage, constant at any one current density. It is assumed that during the time of taking Readings 1, 2, 3, E does not vary. This is proved when Reading 3 checks Reading 1. Results were not recorded unless this was the case.

On trying out the Anodes Nos. 1, 2, 3, 4, 5, 7, which were supplied with sulphur dioxide gas, it was found that with Anodes Nos. 1, 2, 3 there was no reduction in polarization at all. The reason Anode No. 3 was tried was because, from a few random tests with the carbon anode, it had been learned that with this anode there would be beneficial reduction in polarization by the introduction of sulphur dioxide gas. It was thought that charcoal, being quite porous, might promote anode efficiency on account of its known property of occluding gases. A charcoal rod, when used as anode, was found to have too high resistance to permit current to pass, therefore the combined lead-charcoal anode (No. 3) was made. This anode did permit current to pass, but, contrary to expectation, there was no beneficial depolarization.

The carbon anodes Nos. 4 and 5, the former with radial holes and the latter with saw cuts, were tried with their submerged ends of central channel open. Then they were tried with their submerged ends plugged, forcing all the gas entering the electrolyte out through the radial holes and



saw cuts. A variety of arrangements of holes and saw cuts were tried. Finally Anode No. 7, carbon anode with central channel extending through (not plugged) and with no other opening (neither radial drill holes nor saw cuts), was tried. The gas was delivered into the electrolyte entirely

from the lower extremity of the anode. The depolarization efficiencies of Anodes Nos. 4, 5, 7 were equally good compared one with the other. Therefore Anode No. 7, being the simplest in form as well as one of the most efficient, was selected to be used in Test No. 3 and in Test No. 4. Readings were then taken from which Fig. 14 has been drawn. It is to be noted that during this test, sulphur dioxide gas was admitted through every anode, at such a rate that there was a gentle bubbling of the gas in the electrolyte.

Depolarization was not always equally good with the same carbon anode. It was generally only with a new carbon anode, when on closing the circuit permitting current to pass at current density of 20 amperes per square foot, that a total voltage reading as low as 0.60 volt might be recorded. This extremely low voltage always rose to some higher value which remained constant during the test. A subsequent test with the same anode generally gave the first voltage reading higher, likewise, the reading of the constant value higher. When depositing copper with a current density of 20 amperes per square foot, at one stage of the experimenting it appeared that the total voltage could be held at 0.9 volt, at a later stage, 1.2 volts, while Test No. 4 recorded 1.3 total volts. Apparently a new anode, when first put into use, is more efficient in reducing polarization than subsequently. This may partially account for variations in tests on the same anode. From time to time there was considerable variation in the ohmic resistance of cell (due to contacts), which may account for some of this variation in total volts.

Finally, after completing all the regular readings for the curve sheet, a little further experimenting was done. Ten anodes were used in a test. Gas was admitted by the ten anodes, by four anodes only, and by six anodes only. The reduction in polarization was as good when admitting gas by four anodes or by six anodes as when admitting gas by all ten anodes. The only difference noticed was that the gas was required to flow through the anodes admitting gas somewhat faster than before—probably the volume of gas entering was no more.

In Test No. 3, when using different kinds of anodes, beneficial reduction of polarization was only secured when carbon anodes were used, indicating that the nature of the anode is important. When using the carbon anode, equally good results were secured with all forms of the carbon tube (National Carbon Co. tube). It mattered not whether they were perforated by drill holes or saw cuts, for admitting and distributing the gas to the electrolyte, or whether it entered by every anode. All that was necessary was that the anode should be a carbon tube, with sufficient sulphur dioxide gas admitted to the electrolyte in some way. It appears that the quantity of sulphur dioxide gas required is that necessary to keep the electrolyte saturated with the gas.

Solubility of Sulphur Dioxide in Water

H ₂ O Degrees C	20	30	40	50	60	70	80	90	100
SO ₂ per cent. dissolved..	8.6	7.4	6.1	4.9	3.7	2.6	1.7	0.9	0.0

(H. O. Hofman, *General Metallurgy*, p. 880.)

Referring to the Tossizza patent, which states: "I have thought to use . . . insoluble anodes kept in contact with sulphurous acid and to utilize the known depolarization properties of the said sulphurous acid;" "These anodes can be made of carbon." From the above quotations, together with the results obtained in this test, I infer that Tossizza experimented only with carbon anodes. (See Figs. 8, 9, 10, and 14.)

Test No. 4.—The operation of this test differed from Test No. 2 only in that carbon tube anodes were used in place of the lead anodes, and sulphur dioxide gas was introduced into the electrolyte. Were depolarization perfect in operation, then for each equivalent of copper deposited, two equivalents of H₂SO₄ would appear instead of one when sulphur dioxide gas is not used.⁴ In this test, the initial charges of electrolyte to the different steps had the degree of acidity which had been determined suitable for Tests Nos. 1 and 2 where SO₂ gas was not employed. Consequently, when using sulphur dioxide gas as depolarizer, the electrolytes became more acid in proportion to the anode efficiency as the electrolysis progressed. The tabulation of Test No. 4 shows a decided increase in acidity. (See Table I.) When the anode efficiency is known and when the initial electrolyte charged to the cells is made correspondingly more acid, then the electrolyte should remain constant in acid contents. Anode efficiency is discussed fully in Test No. 3. It was there estimated by percentage reduction in polarization rather than by increased acidity, as with a fine electrical instrument this method was thought to be more accurate than analytical methods. By the electrical method, results were recorded instantaneously, while the determination of the sulphuric acid by chemical analysis would require days to secure a series of results.

Referring to the tabulation of Test No. 4, where the average results of readings taken half-hourly are recorded, quite a saving in power in Nos. 1 and 2 Steps of Test No. 4 over Test No. 2 will be seen.

Finishing Step.—The curve sheet of the Finishing Step (Fig. 15) represents the performance of this step of the process in Tests Nos. 2 and 4.

The 5,210 cc. of electrolyte, containing 0.743 per cent. copper and 15.4 per cent. H₂SO₄, with current of 1.648 amperes passing, required theoretically, 19.85 hr. to completely deposit the total copper contents. In these tests, when carried to the 19.85-hr. point, the results were:

⁴ *Transactions of the American Electrochemical Society*, vol. xxv, p. 230 (1914).

Test No. 2 Test No. 4

Copper deposited, per cent. of that supplied to Finishing

Step.....	93.5	90.2
Average current efficiency.....	93.5	90.2

While the extraction is high and current efficiency good in both tests, it would not be desirable to carry the electrolysis to this point (19.85-hr.

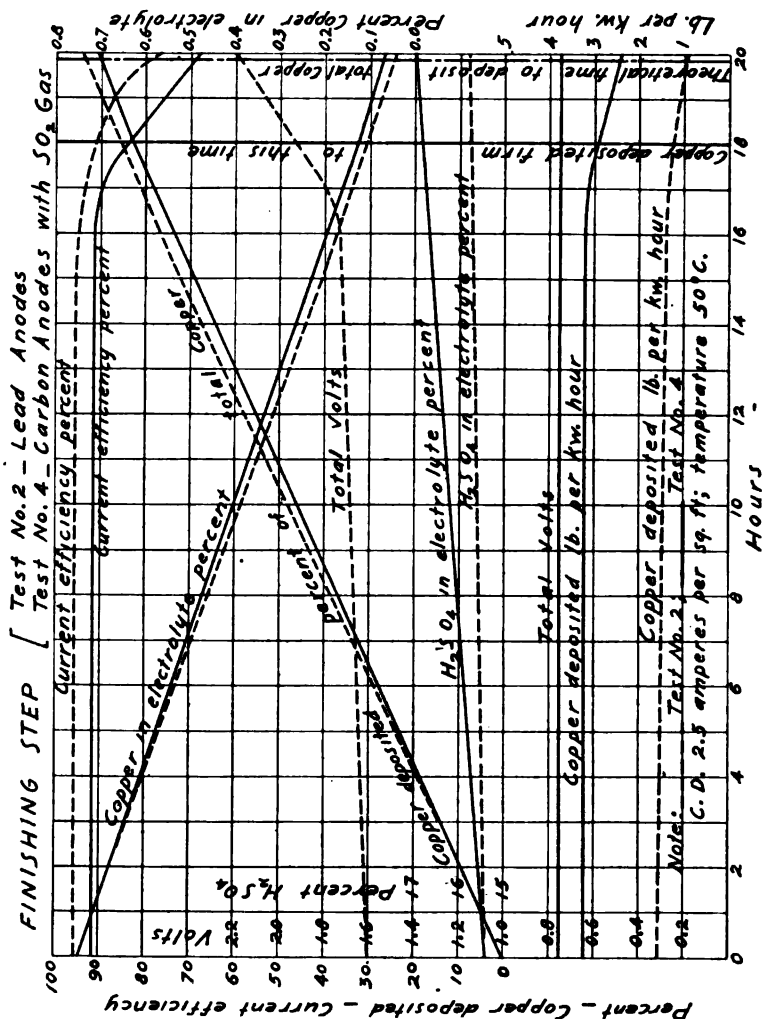


FIG. 15.—FINISHING STEP.

point) in Test No. 2, because the last of the copper comes down spongy and falls to the bottom of the cell. In Test No. 4, the electrolysis may be carried to the 19.85-hr. point and a good cathode produced on account of the presence of SO_2 in the electrolyte. But even in Test No. 4, this point is the limit of feasible electrolysis. At the 20-hr. point, great

blotches of sulphide of copper formed on the cathode. Test No. 2 may be carried to the 18th hour and produce a good, firm deposit of copper. In Test No. 2, when carried to the 18-hr. point, the results were:

Test No. 2

Copper deposited, per cent. of that supplied to Finishing Step.....	85.5
Average current efficiency.....	94.4

Note that there is a decided saving in power by the use of sulphur dioxide gas with the carbon anodes, in the Finishing Step of Test No. 4. In Test No. 4, there is no rise of polarization as the copper contents becomes depleted, while the total voltage becomes 2.2 volts in Test No. 2.

The middle row of Fig. 13 consists of six cathodes. Counting from left to right, the first group of three (Nos. 1, 2, 3) was used in Tests Nos. 1 and 2; the second group of three (Nos. 4, 5, 6) was used in Test No. 4. Immediately below Cathode No. 2, is one of the lead anodes used in Tests Nos. 1 and 2; immediately below Cathode No. 5, is one of the carbon anodes used in Test No. 4. Cathodes Nos. 1 and 4 are from No. 1 Step of their respective tests; Cathodes Nos. 2 and 5 are from No. 2 Step of their respective tests; and Cathodes Nos. 3 and 6 are from the Finishing Step of their respective tests.

The performance in Test No. 4, in pounds of copper deposited per kilowatt-hour—between 1.82 and 3.7 lb.—is not equal to that in Test No. 3. Owing to the construction, electrodes light in weight, and resting on bus-bars, the contact resistance was abnormally high. Special precaution was taken in Test No. 3 largely to eliminate the contact resistance. Even in Test No. 3, the ohmic resistance is to that of practice as 0.108 is to 0.062. Consequently it is believed that even the showing of Test No. 3 can be bettered in practice, and with this plant on repetition.

Test No. 1 showed the performance of lead anodes when used with the step system. Test No. 2 was a modification of Test No. 1, seeking to place the operation on a more practical basis. Test No. 3 compared the performance of different kinds of insoluble anodes with and without sulphur dioxide gas. Test No. 4 showed the performance of carbon tube anodes with sulphur dioxide gas introduced into the electrolyte, when used with the step system. The operation of the Finishing Step with SO_2 gas showed a remarkable saving of power as well as a better deposit of copper over that of the Finishing Step in Test No. 2 with lead anodes and no gas. (See Figs. 9, 13, and 15; Table I.)

Step System of Electrolytic Precipitation.—The copper solution used in the preceding experiments was obtained by dissolving copper sulphate in water. The electrolytic plant was not supplied with copper solution obtained by leaching the ore, because the amount of ore roasted was insufficient to keep the electrolytic plant in operation for the length of time desired. Moreover, it is believed that the only additional informa-

TABLE II.—*Example of Step Arrangement of Plant—Arrangement A*
Copper Contents of the Electrolyte of the Steps held in Geometrical Ratio $\frac{1}{4}$ (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte....	1	$\frac{1}{4}$	$\frac{1}{16}$			$\frac{1}{16}-0$
Copper contents of electrolyte, per cent....	6.0	1.5	0.375			0.375-0
H ₂ SO ₄ contents of electrolyte, per cent....	0	6.93	8.66			8.22-9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		25	6.25			0
Copper deposited, expressed as per cent. of that in feed.....		75	18.75			6.25
			(Nos. 1 and 2 Steps)			
Total copper deposited, expressed as per cent. of that in feed.....		75	93.75			(All Steps) 100
Number of cells in series in each step.....		x	x			x
			4			12
Relative electrode area per cell.....		1	4			8
Relative C. D. in each step.....		1	$\frac{1}{4}$			$\frac{1}{16}$

N. B.—When using carbon anodes with sulphur dioxide gas, the acid contents will be higher than given in these tables.

These tables were worked up, assuming that there was no evaporation of the electrolyte during its passage through the plant, or, what amounts to the same thing, that water was added to the different steps equal in amount to the evaporation occurring in the different steps.

tion that could have been obtained by treating copper solution resulting from ore leaching, would have been the effect of accumulation of impurities in the electrolyte. In order to determine the effect of accumulation of impurities in the electrolyte, and to inaugurate the necessary purification methods, such as chemically purifying the electrolyte or wasting a sufficient amount of barren solution, a greater amount of material than was at hand would have been required, as well as a long campaign of operation. Much information is available regarding the maintaining of the purity of solutions in a cyclic process. The analysis of the ore together with the results of the leaching tests, and the acid consumption, enables one to judge the quantity of impurities entering the leaching solution in the treatment of the ore now under discussion.

The electrolysis of the copper sulphate solution was conducted in steps. All the electrolytic cells were connected in series, while the electrodes of the individual cells were connected in parallel. Each step maintained constant the composition of the electrolyte—both in copper and acid.

The cells comprising each step have two circulations of the electrolyte. There is one circulation, which is designated the step circulation, in which the step circulating pump draws the electrolyte from the last cell of the step, sending the electrolyte to a more elevated tank (temperature tank)

EXAMPLE OF STEP ARRANGEMENT OF PLANT—A.
Copper contents of the electrolyte of the steps held in geometrical ratio $\frac{1}{4}$.

Diagram shows solution circulation only.

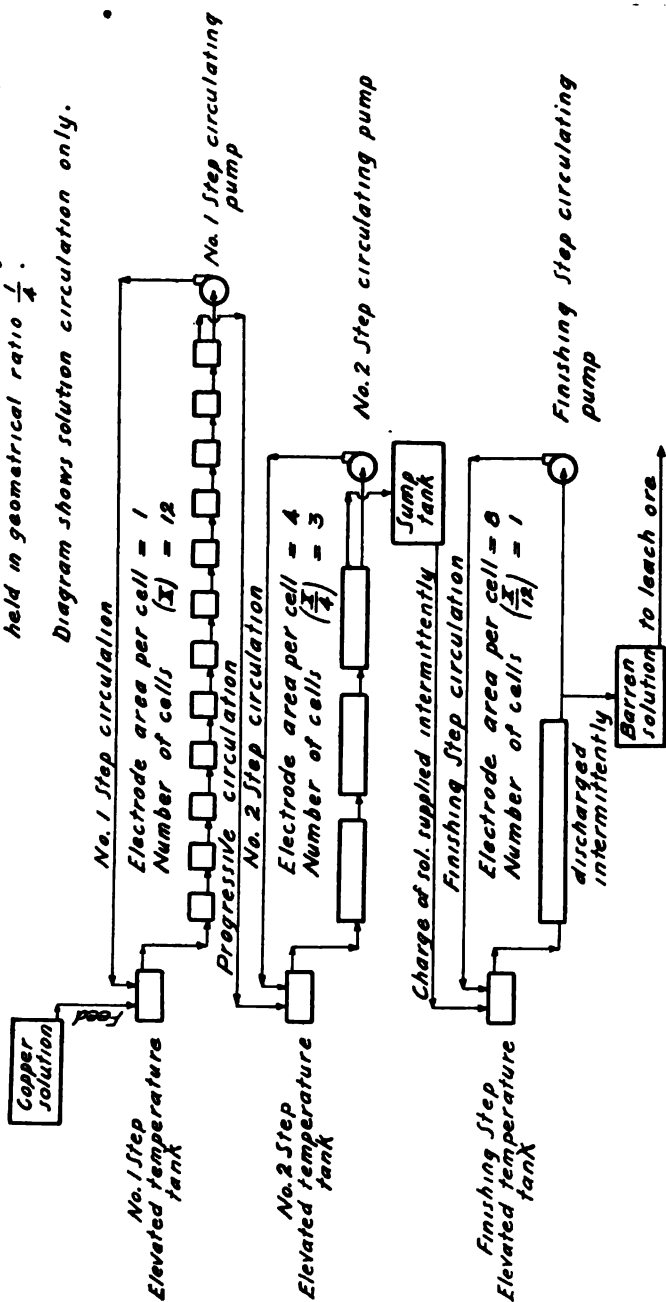


FIG. 16.—STEP ARRANGEMENT OF PLANT—A.

TABLE III.—*Example of Step Arrangement of Plant—Arrangement B*
Copper Contents of the Electrolyte of the Steps held in Geometrical Ratio $\frac{1}{4}$ (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte.....	1	$\frac{1}{4}$	$\frac{1}{4}$			$\frac{1}{4}$ -0
Copper contents of electrolyte, per cent.....	6.0	2.0	0.667			0.667-0
H ₂ SO ₄ contents of electrolyte, per cent.....		6.16	8.22			8.22-9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		33.3	11.1			0
Copper deposited, expressed as per cent. of that in feed.....		66.7	22.2 (Nos. 1 and 3 Steps)			11.1
Total copper deposited, expressed as per cent. of that in feed.....		66.7	88.9			(All Steps) 100
Number of cells in series in each step.....		x	3			x
Relative electrode area per cell.....		1	3			6
Relative C. D. in each step.....		1	$\frac{1}{4}$			$\frac{1}{4}$

in which it may be heated (maintaining the temperature of the system constant), and from which it overflows and gravitates into the first cell of the step. The electrolyte then passes on through the series of cells in the step and finally again to the same circulating pump.

There is the other circulation, which is ordinary progressive movement of the solution through the plant. The copper solution resulting from the leaching of the ore, or otherwise obtained, is run, together with the discharge of the No. 1 Step circulating pump, into the elevated temperature tank of the No. 1 Step. Thus more solution enters the first cell of the step than is drawn away by the circulating pump, consequently an equivalent amount of solution must leave the last cell of the step by way of the overflow discharge. This overflow discharge passes on and joins the discharge of the circulating pump of No. 2 Step, and enters the elevated temperature tank connected with that step. Finally the last cell of the step preceding the Finishing Step, overflows an amount of solution equal in amount to the inflowing copper solution fed to the elevated temperature tank of the No. 1 Step.

The copper contents of the solutions is maintained in geometrical ratio or in arithmetical difference from that of the copper solution fed to No. 1 Step to that of the electrolyte of the step preceding the Finishing Step.

The experimental plant was designed so that a variety of factors could be used. Let us select for a plant a solution resulting from the leaching of the ore with copper contents of 6.0 per cent. and the geometrical ratio $\frac{1}{4}$ for the copper contents of the electrolyte of the different steps. (See Table II and Fig. 16.) Then the electrolyte of No. 1 Step

EXAMPLE OF STEP ARRANGEMENT OF PLANT — B.
 Copper contents of the electrolyte of the steps held in geometrical ratio $\frac{1}{3}$.

Diagram shows solution circulation only.

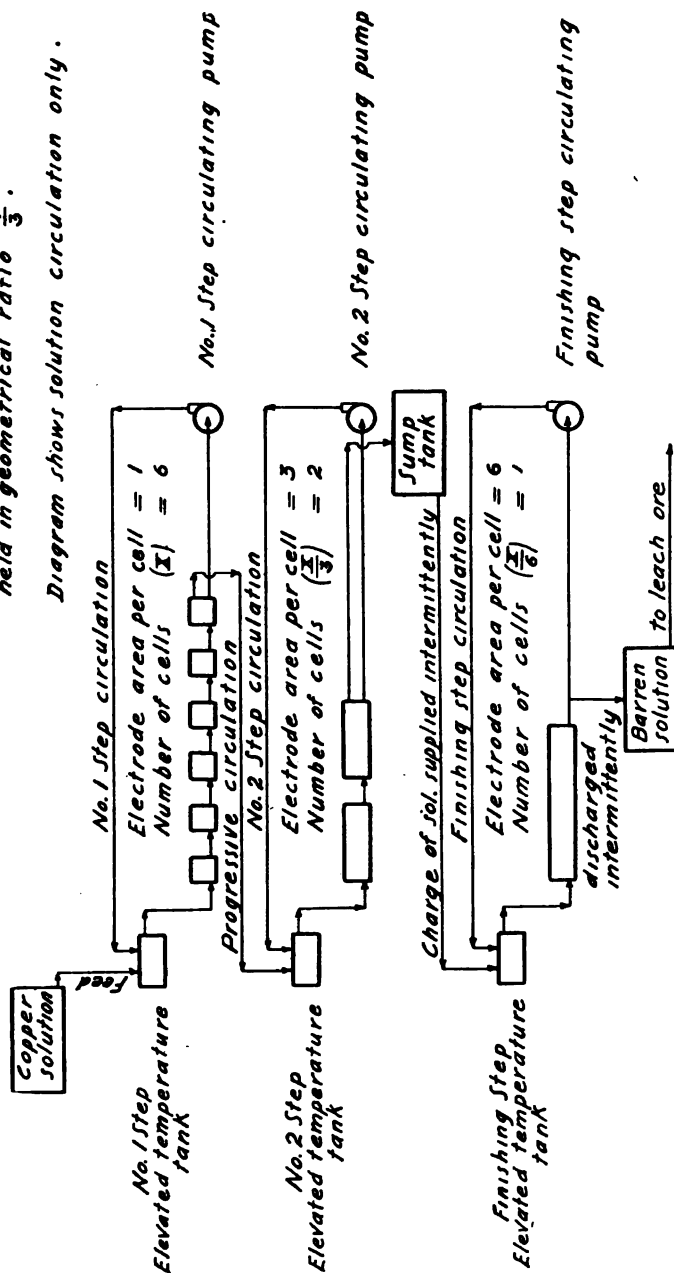


FIG. 17.—STEP ARRANGEMENT OF PLANT—B.

TABLE IV.—*Example of Step Arrangement of Plant—Arrangement C*
Copper Contents of the Electrolyte of the Steps held in Geometrical Ratio $\frac{1}{2}$ (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte....	1	$\frac{1}{4}$	$\frac{1}{4}$	$\frac{1}{4}$	$\frac{1}{16}$	$\frac{1}{16}$ —0
Copper contents of electrolyte, per cent....	6.0	3.0	1.5	0.75	0.375	0.375—0
H ₂ SO ₄ contents of electrolyte, per cent....		4.62	6.93	8.08	8.66	8.66—9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		50	25	12.5	6.25	0
Copper deposited, expressed as per cent. of that in feed.....		50	25	12.5	6.25	6.25
			(Nos. 1 and 2 Steps)	(Nos. 1, 2 and 3 Steps)	(Nos. 1, 2, 3 and 4 Steps)	(All Steps)
Total copper deposited, expressed as per cent. of that in feed.....		50	75	87.5	93.75	100
Number of cells in series in each step.....		x	x	x	x	x
			2	$\frac{1}{2}$	8	8
Relative electrode area per cell.....		1	2	4	8	16
Relative C. D. in each step.....		1	$\frac{1}{2}$	$\frac{1}{4}$	$\frac{1}{16}$	$\frac{1}{16}$

will contain $6.0/4$ equals 1.5 per cent. copper, and 6.93 per cent. H₂SO₄. Let us further assume for No. 1 Step 12 cells in series, each with one unit of area of cathode surface. Next select the most suitable current density for this step. The copper contents of the electrolyte is the governing factor in making this selection, since it is desirable always to maintain current density proportional to copper contents of the electrolyte. The selection of current density fixes the strength of current. The copper solution is fed to this step at such a rate that the copper deposited on the cathode amounts to three-quarters of the entering copper. The electrolyte circulating in the step thus remains constant in copper contents and in acid, and the solution overflows in volume equal to that of the inflowing feed solution.

The solution fed to No. 2 Step, being the overflow of No. 1 Step, contains 1.5 per cent. copper and 6.93 per cent. H₂SO₄. Since the electrolyte of this step contains one-quarter as much copper as that of the preceding step, then in order to maintain the electrolyte of this step constant one-quarter as much copper must be deposited on the cathodes of this step. This is accomplished by placing one-quarter as many cells in series, namely, using three cells. These cells should each have four units of cathode area, that is, a cathode area four times as great as that employed in each cell of No. 1 Step, since in this Step (No. 2) the electrolyte contains one-fourth as much copper, which requires a current density one-fourth that of No. 1 Step to be employed.

It is seen that the solution overflowing from No. 2 Step contains but 6.25 per cent. of the original copper contents. This solution in which the acid has been regenerated may pass on and be used to leach ore. That

it contains a small amount of copper is no detriment since this copper will return in the enriched solution and will not be lost.

In case it should be desired to extract the copper to the last trace, one more step—a finishing step—may be added to the plant. This step would be supplied intermittently with a charge of solution from the sump tank,

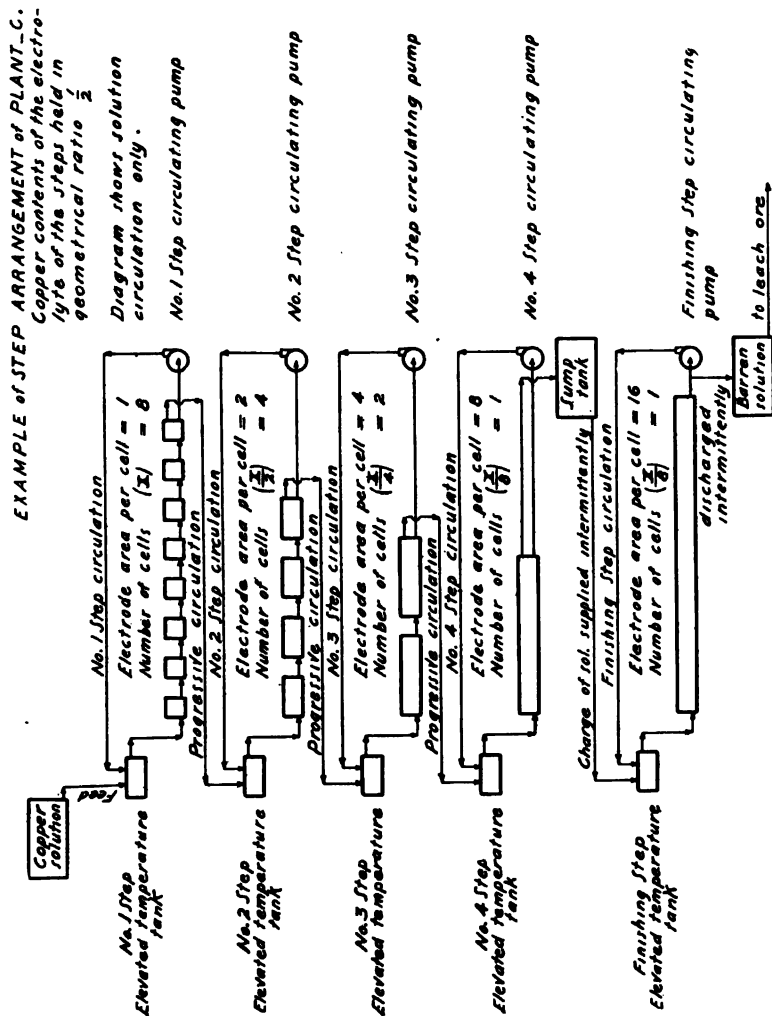


FIG. 18.—STEP ARRANGEMENT OF PLANT—C.

where the solution overflowing from No. 2 Step has collected. This charge of solution would be circulated in the Finishing Step by a circulating pump in the same manner as in the preceding step but with neither feed nor overflow, until all of the copper contents is deposited on the cathode. The solution, barren in copper, would be withdrawn from the system, after which a new charge would be supplied.

TABLE V.—*Example of Step Arrangement of Plant—Arrangement D*
Copper Contents of the Electrolyte of the Steps held in Arithmetical Difference, 1.5 Per Cent. (except the Finishing Step)

	Feed-Copper Solution	No. 1 Step	No. 2 Step	No. 3 Step	No. 4 Step	Finishing Step
Relative copper contents of electrolyte.....						
Copper contents of electrolyte, per cent....	6.0	4.5	3.0	1.5		1.5-0
H ₂ SO ₄ contents of electrolyte, per cent.....		2.31	4.62	6.93		6.93-9.24
Copper remaining in electrolyte, expressed as per cent. of that in feed.....		75	50	25		0
Copper deposited, expressed as per cent. of that in feed.....		25	25	25		25
			(Nos. 1 and 2 Steps)	(Nos. 1, 2 and 3 Steps)		
Total copper deposited, expressed as per cent. of that in feed.....		25	50	75		(All Steps) 100
Number of cells in series in each step.....		x	x	x		x
Relative electrode area per cell.....		1	1.5	3		6
Relative C. D. in each step.....		1	$\frac{3}{4}$	$\frac{1}{4}$		$\frac{1}{6}$

The number of cells in series in the Finishing Step would be one-third the number used in No. 2 Step. Then this step must operate continually to deposit all the copper sent to it by the overflow of No. 2 Step, since the amount of copper contained in the overflow solution of No. 2 Step is one-third the amount deposited in that step. The cells of the Finishing Step should each have (in this example there is but one cell) eight units of cathode area, that is, twice the cathode area and one-half the current density as that employed in each cell of No. 2 Step, as in this step (Finishing Step) the electrolyte has on an average one-half the copper strength.

Even when the Finishing Step is employed, it may be desirable to stop the electrolysis somewhat short of complete extraction. (This matter is discussed in connection with Tests Nos. 2 and 4.)

In order to obtain the figures given above, it is necessary to compensate for evaporation, which at the temperature employed (50° C.) and with the large surface of solution exposed is considerable. In Test No. 1, evaporation was compensated for by the addition of water to the temperature tank of each step, equal in amount to that of the evaporation. Thereby was demonstrated the feasibility of maintaining the composition of the electrolyte constant, both in copper and acid, during the electrolysis, and with copper contents of the electrolyte of the different steps in the above-described ratios (4:1)

In Tests Nos. 2 and 4 the plan of operation was slightly modified from that followed out in Test No. 1 in order to somewhat simplify the operations. To the temperature tank of No. 1 Step was fed a more dilute copper solution, which amounted to the original volume of the 6.0 per cent. copper solution plus the required amount of water to compensate for evaporation of No. 1 Step. Consequently No. 1 Step oper-

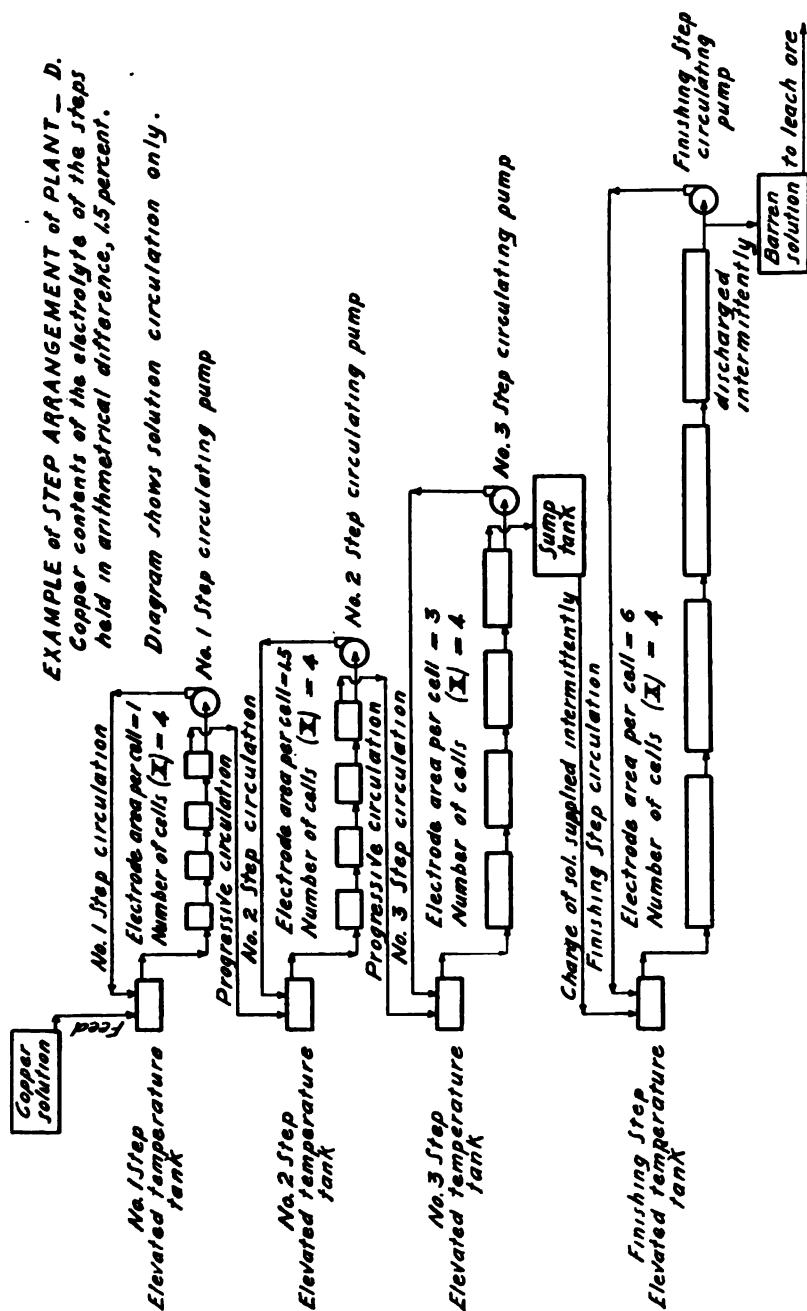


FIG. 19.—STEP ARRANGEMENT OF PLANT—D.

ated in every way exactly as it did in Test No. 1, thus maintaining the composition of its electrolyte the same and causing an overflow equal in amount and composition to that in Test No. 1. This calling for a more dilute copper feed solution might be an advantage in practice, as it might be easier to secure than the one of higher copper contents. Further dilution of the copper solution fed to No. 1 Step, to compensate for evaporation in No. 2 Step, is not permissible as it would derange the constant conditions desirable to be maintained in that step. Moreover, for simplicity of operation, water was not added to No. 2 Step to compensate for evaporation, consequently the copper contents of the electrolyte of No. 2 Step differed from the original geometrical ratio. A higher value of both the copper and the acid contents obtained. Such higher values of both copper and acid, however, remained constant. The higher copper value would permit of a somewhat smaller electrode surface and higher current density per cell than was called for in Test No. 1.

The ideal of the copper hydro-metallurgist is to secure, in the hydro-electrolytic extraction of copper from its ores with insoluble anodes, conditions comparable with those which obtain in the electrolytic refining of copper with soluble anode. By means of the step system it is believed that these conditions are more nearly approached than they have been heretofore. The desirable conditions are: (a) Constant composition of electrolyte with current density adjusted to suit composition; (b) small power consumption.

The step system arrangement of the electrolytic cells accomplishes "a" and also permits the electrolyte to be circulated at any rate desired as in electrolytic refining of copper. A rapid rate of circulation in the individual cells which is made possible by the step system, together with the adjusting of the current density proportionate to the copper contents (which is held constant in each step), makes possible the securing of high current efficiency.

As most copper ores contain sulphides, they should be roasted prior to leaching. Sulphur dioxide gas is evolved. In such cases it may be desirable to pass this gas through the electrolytic cells, utilizing it as a depolarizer and at the same time producing additional sulphuric acid for the leaching. Ordinarily in leaching copper ores, the solution takes up some iron from the ore. Were no sulphur dioxide gas introduced into the cell, the ferrous sulphate would become oxidized to ferric sulphate at the anode. The ferric sulphate formed would be carried by circulation of the electrolyte to the cathode where it would dissolve some of the deposited copper, again becoming ferrous sulphate, after which the cycle would be repeated. Current efficiency would thus be decreased. But the sulphur dioxide gas when employed with a suitable insoluble anode, besides maintaining a high current efficiency, is beneficial in reducing the power

consumption to a point more comparable with that consumed in the electrolytic refining of copper with soluble anode. (See Figs. 14 and 15.)

Increased areas of electrodes to give reduced current density for corresponding depletion in copper contents of the solution in the process of electrolysis have been used heretofore. In these processes, however, there is but one circulation of the electrolyte, namely, that through the plant progressively; the solution becomes depleted of the copper during its passage. Such a circulation would necessarily be slow and insufficient for securing the best results. Moreover, the electrolyte, due to the slow progress through the cells, would vary in composition in different parts. So, although in the ordinary processes the aim is to maintain the current density proportional to the copper contents, it has not really been accomplished.

The step arrangement of the plant is such that—

1. Rapid circulation is maintained by the step circulating system.
2. The composition of electrolyte remains absolutely constant in each step of the plant.
3. The current density is held strictly proportional to the copper contents of the electrolyte.

SUMMARY

It was desired to select an Arizona problem, so the treatment of a porphyry copper ore seemed to be one of the most important. This kind of ore when treated for the extraction of its copper by mechanical concentration and smelting, the methods most generally employed, yields 66 per cent. or less of its copper.

In these experiments, therefore, the aim was to determine a better method of treatment. The method selected for investigation was one in which metallurgists are at present doing much experimenting in the hope of demonstrating the superiority of leaching and electrolytic precipitation over earlier practice, namely: 1, Oxidizing roast; 2, leaching with dilute sulphuric acid; 3, electrolytic precipitation of the dissolved copper.

While it was planned to carry on a complete systematic test of the ore, thereby enabling one to determine the suitability of this ore to the hydro-electrolytic treatment, special stress was laid on roasting and on electrolytic precipitation using sulphur dioxide to lessen the consumption of power and to produce additional sulphuric acid.

It has been demonstrated in these experiments that it is feasible to roast successfully the ore, so that when leached with hot dilute sulphuric acid, the entire copper contents may be obtained in solution.

Some of the practical engineers say that they secure but little beneficial reduction in power consumption when using sulphur dioxide gas as a

depolarizer. Tossizza says, "I have thought to use . . . insoluble anodes kept in contact with sulphurous acid, and thus to utilize the known depolarization properties of the said sulphurous acid. These anodes can be made of carbon, and in this case the sulphurous acid can be introduced outside the anode or in the interior thereof;" "One can thus obtain a very beautiful deposit of pure electrolytic copper . . . with a sufficient intensity at a voltage of about six-tenths of a volt." The experiments reported under Tests Nos. 3 and 4 corroborate Tossizza's statement in that the method of introducing the sulphur dioxide into the electrolyte is unimportant, provided only that it be introduced in sufficient quantity to keep the electrolyte saturated.

Regarding the saving of power (depolarization), these experiments show depolarization by the use of sulphur dioxide only when used in connection with carbon anodes, while Tossizza, although specifically suggesting a carbon anode, intimates that the same may be obtained with other insoluble anodes. Tossizza does not state the current density at which he was operating when depositing copper with the extremely low voltage of six-tenths of a volt, although "with a sufficient intensity" might mean commercial current density. Experiment Test No. 3 shows likewise that copper deposits continuously at six-tenths of a volt when current density equals 5.8 amperes per square foot, depositing 4 lb. of copper per kilowatt-hour. (See Fig. 14.) It is to be noted that although beneficial depolarization and consequent saving of power is secured at all current densities, when SO_2 is introduced into the electrolyte, it is relatively not the same in amount but decreases as the current density increases.

It is to be hoped that future experiments will demonstrate how to secure the same beneficial depolarization with sulphur dioxide when electrolyzing with high current densities. Likewise whether, and how, beneficial depolarization by sulphur dioxide may be secured with other kinds and types of insoluble anodes.

When it is desired to extract the copper from the electrolyte down to a small trace, sulphur dioxide is very beneficial, not only in reducing the power consumption but in causing the copper to deposit more firmly on the cathode, which when sulphur dioxide gas is not used forms as a spongy deposit toward the last of the electrolysis. On attempting to extract the last trace of copper from the electrolyte, sulphide of copper forms on the cathode.

Regarding the lead anodes, there is no depolarization when used with or without sulphur dioxide gas. There was no loss in their weight during the test.

When starting out on this line of investigation, it was found necessary to design and construct a complete plant on a miniature scale. In doing this, a novel idea appeared—the step arrangement—which greatly facili-

tated the work in all stages of the process, especially in the finishing step where the copper was extracted down to a trace with the production of a good firm cathode, when sulphur dioxide gas was used.

Test No. 1 demonstrates, when using the step system, the feasibility of depositing copper continuously from the electrolyte, while at the same time the copper contents of said electrolyte remains undiminished. A plant composed of several steps of such cells may be operated so that a liquor strong in copper flows into the upper step of the plant continuously from the leaching vats at such a rate that a liquor depleted of its copper, in which the acid solvent has been regenerated, outflows from the plant continuously to the leaching plant. The advantages of a step arrangement of plant are apparent in that the electrolysis is conducted under constant invariable conditions in each cell of the respective steps, as well as in every part of the cell. The cathode area of a step may easily be adjusted so that the most desirable current density is secured (as determined by experience in operating). It is to be noted in step arrangement of plant, the electrolyte may be circulated in any of the steps at any desired rate. This has been shown to be beneficial in promoting high current efficiency.

The experiments demonstrate that the porphyry ore, when treated by a hydro-electrolytic process, will yield its entire copper contents as a good grade of cathode copper. It is therefore hoped that by using a hydro-electrolytic method, lower-grade copper porphyry deposits may be worked than formerly could be worked by methods of concentration followed by smelting.

Epitome

1. The most suitable temperature for roasting the Arizona porphyry copper ore, containing sulphides, in order to render it amenable to acid leaching methods, is between 600° C. and 725° C. The more finely ground the material the shorter the time required for the roasting so as to produce the maximum amount of soluble copper: materials which will pass through a 20-mesh screen and remain on 80 mesh require about 2 hr. roasting at 600° C. to 725° C.; and when ground to pass through an 80-mesh screen the time required is about 1¼ hr. If the roasting is concluded at temperatures above 800° C., the oxidized copper is converted into a compound which is insoluble in dilute sulphuric acid. The longer the roasting is conducted above 800° C., the greater the amount of insoluble copper produced.

2. A heated solution is necessary to leach efficiently the copper from the roasted material. A 10 per cent. H_2SO_4 solution at 100° C. leached out, in from 3 to 6 hr., all the copper from all roasted materials (through 40 on 80 mesh, through 80 mesh, whole through 20 mesh), except material through 20 on 40 mesh, in which case the extraction was not so high.

3. The nature of the anode is an important factor in securing depolarization by sulphur dioxide gas.

4. Depolarization with consequent saving in power is accomplished when using sulphur dioxide gas with a carbon anode, while there is no depolarization with a lead anode.

5. The depolarization by sulphur dioxide gas, even with carbon anodes, does not reach the theoretical amount, being between 45 and 65 per cent.

6. The amount of depolarization effected by sulphur dioxide gas when used with carbon anodes varies with the current density, being a maximum at low current density.

7. The method of introducing the sulphur dioxide gas into the cell is unimportant. All that is necessary is that it be introduced in some way, and in such quantity that the electrolyte is saturated with the gas, so that there is some escaping by bubbling at the surface.

8. A smoother deposit of copper forms when using sulphur dioxide gas as a depolarizer than when not using it.

9. In the electrolysis of an acid solution of copper sulphate, when sulphur dioxide is not supplied to the cell and when one is endeavoring to carry electrolysis to the point of complete extraction, a soft spongy deposit begins to form on the cathode before the complete extraction of the copper is effected. There is also a considerable rise in polarization when the copper contents of the electrolyte becomes low.

10. In the electrolysis of an acid solution of copper sulphate, when sulphur dioxide is supplied to the cell, the copper contents of the electrolyte may be reduced to a very small trace with the formation of a good, firm cathode, without rise in polarization toward the end. Current density and energy efficiency remain high to the end. It is only when prolonging the operation beyond the time when but a small trace of copper remains that sulphide of copper forms as a thin coating on the cathode.

11. Lead anodes do not peroxidize or deteriorate appreciably when used with or without the introduction of sulphur dioxide into the electrolyte.

12. A novel idea—step arrangement of process—makes feasible the depositing of copper continuously, while the copper contents of said electrolyte in the respective steps remains constant. A plant composed of several steps of such cells may be operated so that a liquor strong in copper flows to the plant continuously, and the liquor which outflows from the plant is depleted of its copper.

13. The circulation of the electrolyte by the step arrangement increases the current efficiency. Such rapid circulation is not possible in plants as ordinarily arranged.

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The Advantages of High-Lime Slags in the Smelting of Lead Ores

BY S. E. BRETHEERTON,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

DURING the year 1878-79, Anton Eilers, who was then interested in the lead smelting and refining business near Salt Lake City, Utah, made a somewhat radical departure from the regular practice at that time, which was to use but little lime in the slag, with a high percentage of iron. Lime was not only a cheaper flux than iron, but it enabled the metallurgist to make a more siliceous slag, an economical advantage in smelting where the smelting companies had to purchase either iron or lime to neutralize the excess silica, and penalize the ore shippers accordingly. The larger amount of both lime and silica in the slag also made it of lighter specific gravity, and a better separation of the metals and mattes was secured. This change in the formation of slags for lead smelting was brought about by Anton Eilers, prior to the Leadville mining excitement of 1879. We must also give Mr. Eilers credit for the general introduction of the hollow water-cooled cast-iron jacket for the blast furnace.

Again referring to the use of more lime and less iron in the formation of slags, there is no doubt but that it prevents the formation of a great many so-called "sows" in the lead crucible, such as were made in the early history of lead smelting in the Rocky Mountains; these sows consisted of lead, iron, arsenic, sulphur, copper, and other metals, including gold and silver, mixed with unconsumed fuel, and often, after vain attempts to cut them up or melt them to a suitable size for handling, were buried in the slag dumps to get them out of sight.

Another advantage derived from the adoption of high-lime slags when smelting ore containing arsenic is the production of less speiss (a by-product heavier than matte, resulting from careless operation of the furnace, the lead being driven out of the crucible and sows built up). This speiss, while usually not as rich in precious metals as matte made at the same time, is too valuable to be thrown away; but it is difficult to treat, and it was thrown away or piled on the slag dumps in the pioneer days of lead smelting in this country.

About the time of the introduction of the lower iron and higher lime

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slags in lead smelting, sulphide ores were encountered in Utah and Colorado. Matte was made as a by-product in smelting and arsenic gave less trouble. The general supposition was that the arsenic was taken up by the matte and finally lost in the treatment of the matte. No doubt this was true to a great extent and I gave the matte the credit of taking care of the arsenic until I had to do some special lead smelting in Nevada. The ore to be smelted contained an average of about $3\frac{1}{2}$ per cent. lead, $6\frac{1}{2}$ per cent. iron, $2\frac{1}{2}$ per cent. manganese, 7 per cent. zinc, 21 per cent. lime, 11.7 per cent. magnesia, $8\frac{1}{2}$ per cent. silica, 1 per cent. alumina, 3 per cent. arsenic, and only a trace of sulphur, together with a very small amount of ore, which we called our lead ore, from another mine averaging $11\frac{1}{2}$ per cent. lead, 20 per cent. iron, 4 per cent. zinc, 3 per cent. lime, no manganese, no magnesia, 30 per cent. silica, and only a trace of either arsenic or sulphur. From this it can be seen that there was more than enough arsenic to make a large proportion of speiss (especially as we smelted regularly considerable speiss containing gold and silver from some old slag dumps in the neighborhood), and no sulphur to make a matte to absorb it. At the same time, the bulk of the ore being so low in iron and manganese and high in lime and magnesia, I was compelled to make a slag very rich in lime and magnesia. The result was that we produced no speiss or matte, excepting once for a few hours when we fed in some sulphide ore to see what the results would be. During this smelting campaign of a little over one month, we had no trouble except with machinery and from shortage of fuel, which forced us to bank the furnace at one time for 24 hr. The result was \$36,000 worth of argentiferous lead bullion made in a short smelting campaign without a single "freeze up" or "blow out" and no speiss product. The only detail in our practice which differed from the ordinary method of smelting was to cupel our bullion for shipment until it was rich in gold and silver, so as to use the lead as litharge over again in the charge. The fact that we were able to get rid of the arsenic in the ore and speiss from old slag dumps and make a comparatively clean slag led me to believe that the high percentage of lime base in the charge and the lack of iron with which the arsenic could combine, resulted in the volatilization of the arsenic, this action no doubt being assisted by the oxidizing influence of the litharge used. While our gold and silver losses were satisfactorily low, our lead losses were very high, which was to be expected when using the lead over and over again in such an oxidized mixture and keeping the coke consumption as low as possible. Coke was the only reducing agent used.

While making matte in the blast furnace in New Mexico and Arizona from low-grade copper ore containing plenty of sulphur and sometimes as much as 4 per cent. arsenic in the charge, no arsenic was found in the matte, either by our chemist or the purchaser, as it had all been volatilized.

Returning to the subject of lead smelting, I do not know of any important improvement in the composition of slags since lime slags were introduced by Mr. Eilers.

As the oxidized lead ores of the Rocky Mountain region became exhausted and "sulphides" were mined from the deeper mines, both silver and lead losses became greater, as no provision had been made for the proper separation of the matte from the slag at that time. For several years the slag was tapped into the ordinary small slag pots. The losses were very high by reason of the fact that much of the matte was mechanically held in suspension in the slag, the small pots and the quick cooling of the slag preventing proper settling. Further loss was caused by carelessness on the part of the furnacemen, slag being dumped without having been cooled and broken up to save the matte. Then came the introduction of square cast-iron settling boxes on wheels, which are still in use, but much larger and better constructed than at first.

The production of matte containing a little copper (which it generally does if allowed to settle from the slag properly) produces a slag lower in value; at the same time the matte fall, in addition to the lead fall, increases the smelting capacity of the blast furnace, and requires a lower percentage of fuel. The smelting capacity of the lead blast furnace was also increased, sometimes doubled, by increasing the depth of the ore mixture in the furnace and increasing the blast pressure. This increase in depth and blast pressure was not practical as long as the metallurgist held to the idea that part of his fuel must be charcoal, as the charcoal would crush and pack under such a heavy burden, the fine charcoal and high-pressure blast causing "over" or "top" fires in the furnaces, increasing losses by volatilization and burning the fuel before it reached the proper smelting zone of the furnace.

Improvements have also been made in the more economical handling of the ore, permissible only when handling large amounts, such as belt conveyors from crushers to storage bins, automatic dumping cars for feeding the charges into the furnaces, etc.

As the amount of sulphide ore to be smelted in the lead furnace increased, the matte settlers were improved. The sulphide ore was also ground fine and roasted before smelting. The fine roasted ore was found objectionable, causing too much flue dust and packing of the charge in the blast furnace. Briquetting was tried, but was found unsatisfactory and expensive. Then what was known as the Omaha & Grant roasting furnace was designed to roast and then fuse the fine ore into a pasty mass in the same furnace. This made an ideal product for the blast furnace when broken up properly. This method was used for years, but the loss in lead was excessive. The next and one of the most important advances in lead smelting was the introduction of the Huntington-Heberlein process, or "pot roasting," which reduced the cost of roasting and the lead loss, at

the same time producing a suitable product for smelting in the blast furnace. Following this improvement in the metallurgy of lead came the Dwight-Lloyd sintering machine, a cheaper method of doing the same work.

I think I have mentioned the most important improvements in lead smelting for the production of bullion containing gold and silver. Lead as a collector of gold and silver from very rich oxidized ore has its advantages; the loss of gold and silver in the refining of the lead bullion is smaller than in the refining of copper matte, and when the ore contains sufficient lead it is the only process to adopt. Lead smelting requires not less than $12\frac{1}{2}$ per cent. lead in the charge, so that the concentration cannot be greater than eight of ore to one of bullion. If sulphides are to be smelted, the ore requires a comparatively sweet roast, and in the blast furnace three to five times the fuel is required that is necessary in the production of copper matte. In the blast furnace using ores containing copper sulphides as a collector less fuel is required, roasting is not necessary, more siliceous slags can be made, and a much higher concentration is possible by reason of the fact that a much lower percentage of copper is required as a collector. The preliminary removal of the zinc from a lead-zinc ore permits of an improvement in lead metallurgy. This subject has been quite fully covered in a previous paper¹ by me and in articles by F. L. Wilson and others.

The methods of sampling ores to-day are practically the same as those used more than 30 years ago. At large copper-smelting plants they have adopted automatic samplers to cut expense where the ore not only comes from their own mines, but is comparatively low grade. In custom plants where small lots of extremely rich ore are sampled and followed by samples taken from large lots of low-grade ore, automatic sampling machinery is not reliable. Recently I had an opportunity to observe the sampling of a large lot of rich ore at the Selby Smelting & Lead Co.'s works and I could not ask for a better system of sampling. I will not attempt to describe the method here except to say that it was not done automatically and was expensive, but absolutely fair to both the shipper and the company.

I have not touched on the subject of lead refining for the reason that in the West we are more interested in the production of lead bullion containing the precious metals. The bullion can be shipped to refiners in the East to better advantage than to refine it here and ship the metals separately.

In conclusion I wish to call attention to the following items in this article:

¹ The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process, *Bulletin* No. 91, July, 1914, pp. 1771 to 1777.

1. The advantage of high-lime and low-iron slags, especially when smelting lead ore containing arsenic.

2. The advantage of enriching the lead bullion by cupelling so as to use the lead over again as a collector of the precious metals; also the saving in freight on lead in isolated districts where the freight costs amount to nearly as much as the lead is worth.

The cupelling furnace referred to is such as I used in Leadville, Colo., and in Nevada, large enough to cupel 8 tons of bullion in 24 hr. It would also be suitable for the saving of antimony, thus shipping cleaner lead.

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Mining Conditions on the Witwatersrand

BY W. L. HONNOLD, JOHANNESBURG, SOUTH AFRICA

(San Francisco Meeting, September, 1915)

OWING to a unique labor situation and other unusual circumstances, the mining methods of the Rand are hardly comparable with practice elsewhere. They are of considerable interest, however, and their importance is increased by the magnitude of the industry with which they are identified, gold of \$166,186,000 value, say 40 per cent. of the world's production, having been recovered from 25,701,954 tons of ore milled in 1914.

In the sketchy remarks to follow, attention will first be drawn to some influential factors of general bearing, after which certain features at two mines of the writer's group will be referred to. It is hoped that engineers associated with similar or different conditions and practice will contribute to broaden the discussion.

Before proceeding, recognition is due the generous assistance of C. E. Knecht and F. A. Unger.

GENERAL OBSERVATIONS

The principal mines are situated along the northern limit of an irregularly shaped geological basin, and extend in almost unbroken continuity over a distance of about 60 miles. As the boundaries are defined by vertical planes there are both "outcrop" and "deep-level" properties, all differing in size and in technical features.

So much has been written in regard to the geology of the Rand that only a few pertinent features need here be recalled.

Briefly, the ore deposits are profitably enriched portions of certain conglomerate beds or "reefs," rarely more than three, which are interposed between quartzite, or quartzite and slate, and lie so closely together that where more than one are mined they are dealt with as a working unit, although they may be stoped separately. Their collective extension is locally referred to as "the Reef."

Along the outcrop the series dips more or less steeply, and in depth it flattens, the latter tendency being most marked in the Far East Rand, where the bottom of the basin is nowhere so deep as to be inaccessible.

Dikes and faults are a common feature throughout. They are,

however, so well understood as seldom to involve mistaken procedure, although they may occasion delay and added expense.

The pebbles are mostly of rounded quartz, averaging perhaps less than an inch in diameter, cemented together by a fine-grained quartzitic matrix of metamorphic character. Sedimentary inconstancies, including partings of quartzite, are in evidence, and more or less pyrite is interspersed, which may or may not be appreciably auriferous. Native gold occurs in varying amount, constituting on the average perhaps more than half the assay value of the orebodies.

In the matter of thickness the reefs show distinct local irregularity and differ as to individual tendency and as to the common tendency at particular points. In places they are only a few inches thick, sometimes narrowing to a scarcely perceptible seam with or without scattered pebbles, while in other places they exceed a thickness of 10 ft., a rough average being, say, 33 in., which may or may not include quartzite partings. Thickness, however, is of secondary importance and may be offset by assay value. Hence the common adoption of the "inch-pennyweight" as a unit of measurement, from which the still more significant factor of "value over stope width" may be derived in accordance with the apparent breaking conditions of the locality.

The ore deposits separately considered vary as to occurrence, structural features, and internal and relative enrichment. Along the outcrop the more favored mines have practically no barren territory; in fact, in the Central Rand, where enrichment is unusually persistent, the "leader" has been almost continuously stoped over a length of about 5 miles in the upper workings of several adjoining mines, and on the dip more or less to about 7,000 ft., the deepest development being of a reassuring nature, as will be clear from the subsequent reference to the Village Deep mine. At other mines, and characteristically in certain districts, the bodies, although smaller and erratically disposed, are large enough to be stoped separately, and are so frequently recurrent as to insure fairly stable working conditions. There are also places where considerable areas are rendered profitable through the prevalence of interspersed minor points of enrichment; in other words, the reef is "patchy" but can be worked to advantage as a whole. Somewhat similar irregularity is observable in depth and, although local circumstances may justify considerable confidence, there is always a measure of uncertainty as to the outcome of deeper development at any particular point.

Apart from the irregularity last referred to, it is noticeable in the deeper workings of to-day that the ore is largely confined to a single reef and that where the conventional series is present the main-reef leader is the more persistently enriched. A corresponding tendency is also revealed in the Far East Rand, where the gold is associated with a somewhat different geological sequence, several reefs being more or less profitably

enriched near the outcrop but only one to a considerable extent in the deeper mines.

As to enrichment in depth various views have been expressed, influenced perhaps by dissimilar experience, and reflecting probably different standards of comparison. It seems sufficient in the circumstances merely to state that, in the writer's opinion, the average quantity of gold per unit of reef area lessens in depth and the ore deposits show more instances of lower than of higher grade. Decreasing average grade, however, is not the only factor involved, nor is it of the relative importance usually assumed, as will be inferred from the following references to deep development in the two more prominent districts.

The Village Deep mine, located in the Central Rand, has at a horizontal distance of about 7,000 ft. from the outcrop reached a vertical depth of 4,600 ft., the dip in the lower levels averaging nearly 40°, with a steepening tendency. Although the tenor of enrichment is inferior to that of mines nearer the outcrop and the South Reef is of decreased importance, the development on the leader in the lower levels still discloses practically continuous ore, of good stoping width and of better than the average grade of the mine, for over 5,000 ft. along the strike, the full extent in this direction, as well as in depth, being still undetermined.

In the Far East Rand, Brakpan Mines has reached a vertical depth of 4,100 ft. at a distance of nearly 4 miles from the outcrop. The formation, therefore, has flattened, local dip averaging about 7°, and the tonnage per claim being accordingly reduced. Only the contact reef, lying at the base of the series on a slate foot wall, is of material importance, and of its total tonnage about 50 per cent. has so far been of profitable grade. The orebodies are clearly segregated, irregular as to occurrence and shape, generally elongated in the direction of dip, of good thickness, and large enough to be stoped separately, some measuring over 700 ft. along the strike and more than 4,000 ft. on dip. Although they vary in the matter of enrichment, both internally and comparatively, and show a general tenor below that of neighboring mines nearer the outcrop, yet their average is approximately that of the Rand of to-day.

An even more interesting case is that of Springs Mines in the same district, since its workings are 3½ miles from those of its nearest neighbor and about 7 miles from the outcrop, the vertical depth being 3,750 ft. The mine is still in the development stage, but has been laid out on such broad lines as to make the showing of considerable significance. So far, the reef has been opened over an area of about 60 acres, the maximum dip and strike dimensions being respectively 6,200 and 2,500 ft. Characteristics similar to those at Brakpan are disclosed, except that the dip is flatter and that the grade of the ore is exceptionally high, the ore reserve now averaging 10.3 dwt. over an average assumed stoping width of 52 in.

The two last-named mines are specially significant because of their

bearing on the district offering the greatest scope for new activity. It is, however, because of their bearing on the question of ore deposition at great distance from the outcrop, and their influence on mining methods, that they are now referred to. And it is gratifying to find that, although they may afford less profit per unit of area than some of their neighbors nearer the outcrop, they disclose sound mining conditions and promise to justify the heavy capital expenditure involved. Obviously it is imperative in such circumstances that the surface area be ample and not overvalued, that the shafts be so disposed and the layout so conceived as to insure rapid opening in all directions, and that the available capital be sufficient to carry the venture through an extreme phase of poor development or other initial misfortune.

Although the three mines mentioned may not accurately reflect all the available evidence as to enrichment in depth, they serve to elucidate the opinion previously expressed and to suggest that the future of the Rand is perhaps more dependent on the quantity than the quality of the ore yet to be developed.

The reefs generally, as well as the associated quartzite in somewhat less degree, are above the average of vein quartz in hardness, and the formation is notable also for its cohesion and tenacity. In depth these characteristics are partly neutralized by increasing pressure. It is probable, however, that any consequent advantage to stoping is more than offset by the accentuated weakening influence of dikes and faults and the decreased labor efficiency due to higher temperature and greater humidity. There is, as a rule, but little underground water, and, on the whole, working conditions are comparatively favorable.

Some outstanding correlative factors will now be referred to.

Imported supplies are expensive owing to the high rates of transportation, particularly railage. Imported timber costs about \$1 per cubic foot, drill steel about \$150 per ton, and other imported items are proportionately costly. Explosives, locally manufactured of imported material, are on the basis of \$7.33 per case for gelignite containing 60 per cent. of nitroglycerin and \$10.33 for 92 per cent. blasting gelatin. For inferior mine timber of local growth about 32c. per cubic foot is paid. Water for industrial purposes is pumped to some extent from the mines, but mainly from surface dams at small cost, and in exceptional cases is purchased from the Rand Water Board at 80c. per 1,000 gal., which is also the rate for the domestic supply.

In the matter of power, the Rand, apart from the cost of plant, is fortunate. Local coal of about 12.5 lb. bomb test is delivered to individual mines at an average of about \$2.80 per ton, and to centralized power stations at an even cheaper rate. The latter are important factors in the situation, as is shown by the fact that the principal company

is now distributing about 460,000,000 units of electric power over 224 miles of primary transmission line, and 120,000,000 units of compressed air through 30 miles of primary piping. Its charges are on the basis of electric power and average about 1c. per kilowatt-hour. Consequently, the cost of hoisting, pumping, machine drilling, and ventilation is fairly low.

The importance of ventilation and the allaying of mine dust by means of spraying devices has become increasingly apparent in recent years, and the quality of the underground air is now much better than formerly. It is hoped that the measures adopted, coupled with the wider use of hollow-steel drills and greater precaution on the part of employees, will materially improve the position as to phthisis, the economic significance of which is indicated by the fact that about \$3,000,000 per annum is now paid in compensation and about \$50,000 for the maintenance of a sanatorium.

Only a few words seem called for on the subject of labor. Broadly speaking, white employees do but little actual work, their function, which they jealously guard, being to supervise the tasks of the natives. The latter are of various African tribes, differing more or less as to minor characteristics. Generally, they are tractable and fairly dependable, but deficient in dexterity and initiative. They are averse to continuous employment and come to the mines for comparatively short periods, the average stay being about seven months. Hence their progress as workmen is not sustained and standards of efficiency are low. Some, of course, are of exceptional character and capacity and, as a whole, they may in time show greater proficiency than is so far evident. It has to be borne in mind, however, that white labor is unsympathetic to native advancement and that owing to this restrictive influence radical change in labor conditions is improbable, or at least so speculative a matter as to be beyond the scope of the present paper. The relationship between the two classes of labor will be sufficiently apparent from the context. Underground wages vary for different classes of work but average about \$5 per day for whites and 50c. per day for natives. To the latter rate has to be added about 25c. per day to cover recruiting, transportation, housing, food, and medical attention.

BRAKPAN MINES, LTD.

The reef characteristics and significant bearing of this mine have already been mentioned. When first considered its remoteness from known conditions rendered it a speculative venture and, even when \$188,000 had been expended on boreholes and other preliminaries, the outcome was problematical. Thereafter a further expenditure of \$5,772,000 distributed over 8½ years, with consequent deferment of

interest, was called for before the commencement of milling. Perhaps no more notable instance of the confidence which the Rand has inspired is available and, in the light of tendencies elsewhere disclosed, it is doubtful if there has been an issue of more vital importance to the industry.

The dimensions of the property are about 6,000 by 14,000 ft. and the shafts are located as symmetrically as considerations of depth and ventilation would allow. Reference to Fig. 1 will show that the south (No. 2) shaft is centrally situated and that it is to the dip of the north or upcast shaft. This disposition is favorable to rapid development, ventilation, pumping, and underground work generally.

Each shaft has seven compartments and measures 42 by 9 ft. over all, the large dimensions being due to ventilation rather than to hoisting considerations. In the north shaft the reef was reached at a vertical depth of 3,098 ft., and in the south shaft at 3,707 ft., both being continued to provide for bin and loading facilities, as well as tail-rope sheaves. Full particulars of the shafts and of the methods adopted in sinking, also cost details, were given in a lecture delivered by Charles B. Brodigan at the Royal School of Mines, London, on Jan. 27, 1913, and published by the *Mining and Scientific Press*, under the title *Rand Practice in Deep Shaft Sinking* (Apr. 26 and May 3, 1913).

Both shafts are provided with steel headgears 105 ft. high, the combined bin capacity of the latter being about 3,700 tons. The ore is transported by means of a locomotive in 40-ton railway trucks from the north to the south shaft, a distance of 4,300 ft., where it is discharged on to belts which deliver it to the sorting and crushing plant. The connecting railway, which is of standard gauge, extends to the shops, stores, etc., and also to the main line.

Hoisting equipment is installed for only two compartments of the north shaft. The drums, measuring 11 ft. diameter by 8 ft. face, are directly driven by a 1,450-hp. motor with Ward-Leonard control, 2,100 hp. being absorbed in starting. Main and tail ropes are of $1\frac{1}{2}$ in. diameter and the skips carry a net load of 5 tons.

The south shaft has four compartments equipped for hoisting. Two are served by a Ward-Leonard set identical with that at the north shaft and are used solely for the handling of men, tools, and supplies. The cages have three decks and accommodate 60 men each, about 2 hr. being required for lowering or hoisting the day shift of about 110 white and 1,650 native employees. The other two compartments are practically restricted to the handling of ore. Load, ropes, and drums are unaltered, the drums in this case, however, being directly driven by a three-phase 2,100-volt motor of 1,450 hp., using the standard type of liquid controller. Post brakes operated by deadweight and relieved by air pressure are provided in each instance, together with devices for the automatic prevention of overwinding and tooth clutches for adjustment.

About 150 tons per hour are handled over a vertical distance of 3,850 ft., and the cost per ton, as in the case of the Ward-Leonard at the north shaft, averages 17c. when hoisting a total of about 64,000 tons per month.

The arrangement of the station at the bottom of the south shaft will be clear from Fig. 2. It will be noticed that after the trucks are knocked off the various haulages they pass in one direction over the bins, where they are emptied by means of hand-operated revolving tipples. The grade of the station roadways is 1 in 60 and of the curves 1 in 50, the trucks traveling with but slight assistance. The bins, which discharge into Kimberley loading chutes, do not call for description. Nor is it necessary to refer to the arrangements at the bottom of the north shaft, except to mention that an 84-in. Sirocco fan is installed, together with doors and other requirements of ventilation.

Development

Fig. 1 shows the extent to which the mine had been opened at the end of 1914, and makes it clear that the main incline was a matter of first importance. The average dip of the section between the shafts is 7° , its length is 4,500 ft., and the width about 18 ft. This section was driven from both ends and was connected at a point 3,173 ft. from the north shaft, the horizontal error being 6.5 in. and the vertical 0.7 in. Almost throughout, the reef was above the incline, but never more than 40 ft. Stations were cut at intervals of 500 ft. down to the 7th level; below that point, owing to the flatter dip, they were spaced about 800 ft. apart.

It will be recognized that in the development of flat-reef mines such as Brakpan certain features assume a more serious aspect than where the dip is steeper. For instance, the accentuated influences of minor undulations and dislocations of reef render it difficult to avoid excessive sinuosity in driving, which not only adds considerably to development costs by reducing the effectiveness of footage but also interferes with efficient tramming later on. And, obviously, comparatively small faults exert an exaggerated bearing on the length of crosscuts. To meet obstacles of this nature straight-line work is often resorted to, although even this is not free from difficulty if the reef is so disturbed as to require frequent changes in the grade, for the removal of broken ground then becomes more laborious, winches having to be placed at the high points, and the handling of water is complicated by the need of pumps at the low points. The latter handicaps, of course, may also be encountered in pilot drives and haulage-ways, but as to these there is no alternative, since their expeditious advancement, as previously pointed out, is a matter of vital importance in such a mine. It should perhaps be added that the bad

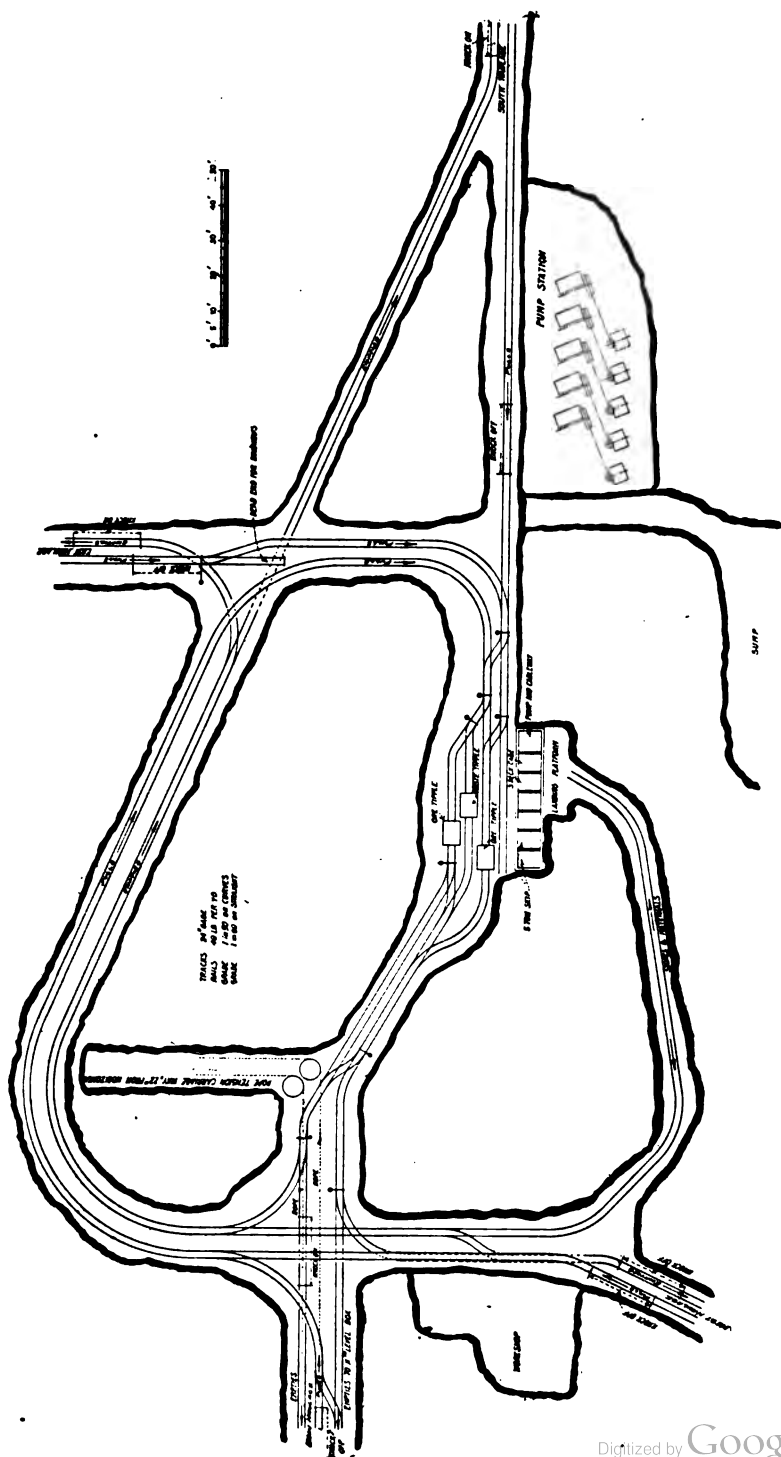


FIG. 2.—ARRANGEMENT OF STATION AND TRACKS, NO. 2 SHAFT OF BRAKPAN MINES.

features referred to largely disappear when the roadways are finally completed and equipped.

The details of development work do not differ materially from common practice. Headings are driven on reef where conditions permit, faults and undulations, however, calling for many departures from the rule. Levels are about 6 by 7 ft.; winzes are carried 9 ft. wide in order to expedite the opening of stopes; main haulageways are 14 ft. wide, and those of secondary importance 11 ft.

Large machine drills are used, mostly of the reciprocating type, with $3\frac{1}{4}$ -in. cylinders. These, however, are gradually being replaced by large hammer-type machines using hollow steel through which water is fed to the bottom of the hole. A high-power explosive is called for and it is found that blasting gelatin, containing 92 per cent. nitroglycerin, gives the best result. Up to the end of 1914 the practice in single-width headings was to blast the cut first, then to turn on a water spray close to the face for about 30 min. to eliminate dust and fumes, and finally to blast the round, all in the same shift and as soon as possible without undue risk to health. In the wider headings, cut and round were blasted simultaneously. Recently the government regulations were so amended as to make it illegal to blast the cut and round separately in the same shift except under special conditions and precautions against dust and fumes approved of by an inspector of mines. Consequently, as far as possible, two single-width headings are now assigned to each miner and the cut and round are blasted on alternate shifts. The full effect of this procedure on costs and the rate of progress is not yet clear.

Drives are equipped with single track of 20-lb. rails laid to 24-in. gauge, and haulageways with double track of 30-lb. rails as well as the necessary endless-rope fittings. Both are electrically lighted and provided with air and water pipes of ample size.

The cost of development during 1914, exclusive of shoveling, tramming, winding, and pumping, but inclusive of equipment, amounted to \$15.87 per foot for drives, crosscuts, raises, and winzes, and \$25.60 per foot for haulageways.

Stoping

Whether hand or machine stoping, or a combination of both, should be adopted in flat-lying mines is a question on which opinions may differ. It is not entirely a matter of natural conditions, since certain associated factors have to be taken into account. In favor of machine stoping it may be urged: (1) that the generally existent shortage of native labor calls for the adoption of mechanical means as far as possible; (2) that hand-stoping natives show less efficiency in such a mine as Brakpan because of their aversion to the "uppers" or dry holes frequently required near the roof; (3) that in most stopes the reef is of fairly good

thickness and its value therefore does not suffer excessively through the increased stope width inevitable with machine mining; (4) that the cost of machine mining is lower, particularly in the wider stopes referred to; and (5) that it will probably enable the supply of a larger average tonnage to the mill. Advocates of hand stoping may claim: (1) that the system requires smaller capital expenditure; (2) that a somewhat narrower stoping width is obtained; and (3) that there is less shock to the roof and consequently less demand for support. Where both methods are in use it is, of course, possible to vary their proportionate employment according to the ebb and flow of native labor and thereby to render the output somewhat less fluctuating.

At Brakpan the unusual course of all-machine mining was decided upon and an equipment of 375 drills capacity is now installed. That the decision was warranted is shown by the fact that the mine has seldom had its full quota of native labor, notwithstanding that the total number of machines on the Rand has since increased by more than 50 per cent. Operations are more sensitive to labor fluctuations than in mines that are partly hand stoped, but this is a matter of secondary importance and, in some measure, is regulated by increasing or decreasing the accumulation of broken ore in the stopes.

Reciprocating machines of $2\frac{3}{4}$ -in. and $3\frac{1}{4}$ -in. cylinder diameter are now mainly used in stoping, the smaller size being restricted to stopes under 60 in. and accounting for about 22 per cent. of the tonnage broken. Small drills of the hammer type with automatic feed and hollow steel are, however, being tried with marked success and promise in time to replace the reciprocating machines throughout. Two natives are attached to each machine—a hammer boy and a spanner boy—and there is a native drill carrier for every two machines. They work under the supervision of a white miner who is on contract and who, as a rule, has charge of four machines irrespective of size. Large machines show an average efficiency of 0.62 square fathom, equivalent to 11 tons, broken per machine shift, and small ones 0.49 square fathom, or 7 tons, the average stope widths being 70 in. and 57 in. respectively. The poor showing, as compared with that obtained elsewhere, is due to labor conventions, white and native, which restrict the daily task per machine, whether large or small, to approximately four holes of 6 ft. depth. The higher efficiency of the large machines may therefore be taken as a measure of the more favorable breaking conditions in the wider stopes.

About 57 per cent. of the explosive used in stoping is blasting gelatin, containing 92 per cent. nitroglycerin, the remainder being gelignite, containing 60 per cent. nitroglycerin. Generally the stronger explosive is used in the narrower stopes and the weaker in the wider ones; individual stopes, however, may call for a reversal of the rule.

The procedure in breaking ore is influenced by the dust factor and the

consequent importance, as well as the advantages, of drilling wet holes. It is therefore the practice when using machines of the reciprocating type to resort to underhand stoping and, when using hammer drills with hollow steel, to back stoping. In opening a stope, work is usually started at both the bottom and top of the winze, which, as already noted, is 9 ft. in width. A winch is placed at the top to deal with the upper portion of the stope, and another half way down to raise and lower the trucks that serve the bottom section by way of the level below. The benches at first conform to diagonals from the winze to the levels, giving the appearance of underhand stoping at the top of the winze and overhand at the bottom, but after reaching a distance of about 100 ft. on either side their lateral extension is checked and they are advanced mainly in the direction of the winze until the stope is opened throughout. Each face is then more or less parallel to the winch track, from which laterals have been thrown off at intervals of 35 ft. for extension as required. "Backs" vary from 300 to 1,200 ft., according to the influence of dip on the distance between levels. Stope pillars of good size are left where necessary, especially along fault lines, and these, together with frequently placed "pig-sty" packs, sufficiently support the roof. As the faces advance right and left the roof of the interior gradually subsides, the weight being taken by the packs. This may or may not call for the shifting of winch tracks. Serious falls of roof seldom occur and the isolation of the orebodies precludes regional movement and therefore renders sand filling or other permanent support unnecessary. After a stope has been mined out and cleaned, unless there are special reasons for keeping it open, the pillars are drawn, a large proportion of the pig-sty poles are recovered, and the roof is allowed to completely collapse. Rock packs are seldom resorted to in the stopes but, in combination with rock walls, are generally employed to protect levels and haulageways that may have to be maintained through worked-out stopes.

Fig. 3 shows one face of a stope that has been well opened, including the track arrangement to be subsequently referred to.

The cost of stoping during 1914 was, for large machines 98c. per ton broken, and for small machines \$1.20. After allowance for the included ore from development faces and other sources, and for the waste rock discarded in sorting, the general average on a per-ton-milled basis was \$1.03.

Shoveling, Tramming, and Underground Haulage

This subject may be divided into two stages. The first covers necessary handling at the working face, hand tramming to the winch track, hoisting or lowering in trains to a level, and hand tramming to a haulageway. The second covers delivery by means of endless-rope haulage to a shaft bin.

Fig. 3, which has previously been referred to, shows the general arrangement of tracks and the position of winches in a stope that has been so far advanced as to call for the shifting of winch tracks. Explanation seems unnecessary.

Rigid trucks, of low design and adapted to the tipples, are used. They hold $20\frac{1}{2}$ cu. ft., equivalent to one ton of ore, have a height of 2 ft. $11\frac{3}{4}$ in. from top of rail, and a length over buffers of 5 ft. 3 in. The truck body is of steel and the bottom is curved to a 17-in. radius. Bottom and ends are of $\frac{3}{16}$ -in. plates and sides of $\frac{1}{8}$ in., a stiffening ring 2 by $\frac{1}{2}$ in.

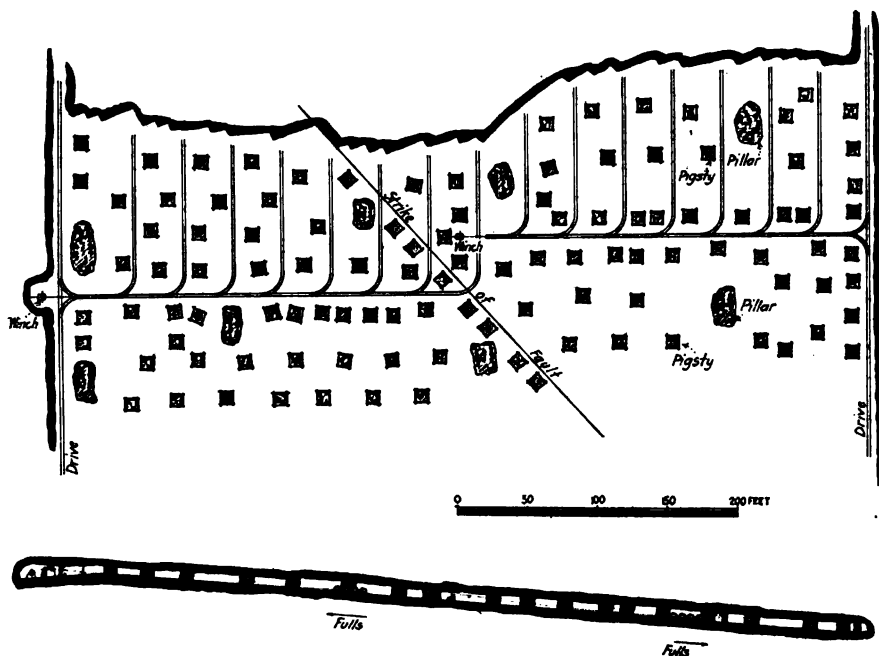


FIG. 3.—IDEAL PLAN OF STOPE, SHOWING GENERAL ARRANGEMENT OF TRACKS, ETC., BRAKPAN MINES.

running around the top and greatly adding to the general rigidity of the body. Brackets, suitable for the Cradock type of overhead clip, are riveted to the ends, which are stiffened by a $\frac{3}{16}$ -in. plate on the inside. The underframe is built of 6 by 4 in. oak, cut to fit the cylindrical bottom of the body, to which it is securely bolted. Cast-steel wheels of 15-in. diameter are used, one being fixed and the other running loose on a 2-in. steel axle which is fitted with a cast-steel grease sleeve. The wheel base is 18 in. and the gauge, as previously stated, 24 in.

Of the natives working on shoveling and tramming, 43 per cent. are loading boys employed at the face. They are on piecework and receive from 8c. to 10c. per truck, which includes loading and delivery to the

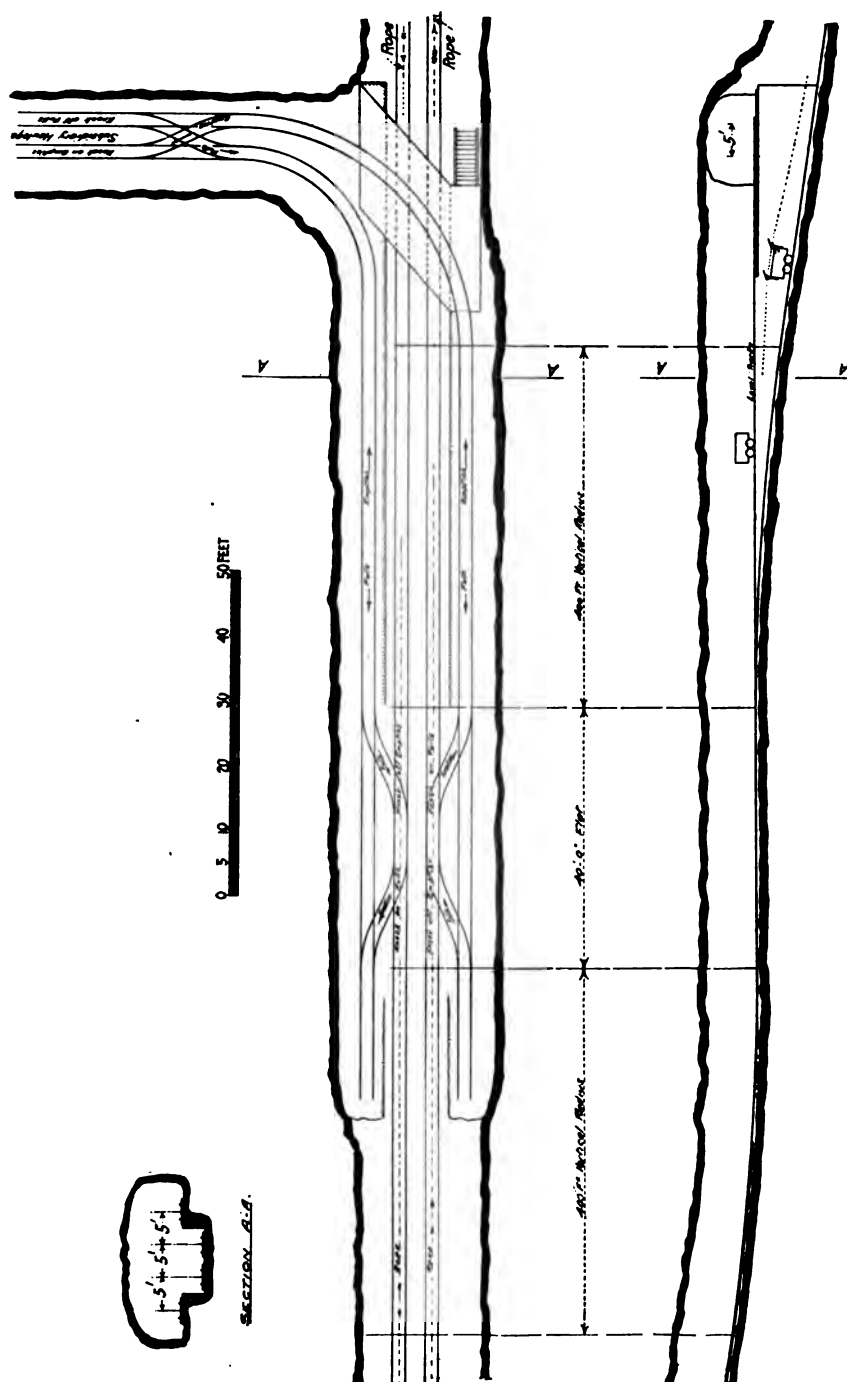


FIG. 5.—ARRANGEMENT SHOWING JUNCTION OF SUBSIDIARY HAULAGE WITH MAIN INCLINE, BRAKSPAN MINES.

All the haulages, as previously mentioned, are provided with double tracks of 30-lb. rails, laid on timber sleepers to 24-in. gauge, and the necessary endless-rope equipment. The gears are electrically driven and designed for a rope speed of 2 miles per hour, at which rate little trouble is experienced from runaway trucks or other mishaps.

The cost of operating all haulages in 1914, during which period a total of 705,775 tons was delivered to the shaft bins, averaged 10c. per ton, about 75 per cent. of the amount representing white and native labor, 5 per cent. power, and the remainder ropes, fittings, workshops, and sundry stores. This also covered the transportation of tools and supplies. Obviously, in comparatively considering the problem of underground haulage, it is necessary to take into account the mileage involved, as well as hoisting and surface transport.

Ore Reserve and Grade Control

Stopes, as well as development, are regularly sampled in detail, and at the beginning of the year the ore reserve is computed. Previously included "blocks" are then recalculated, if their faces have been altered by stoping or further development, and the new tonnage is taken in at its estimated value.

Owing to the elongated form of the orebodies a block between adjacent levels can be estimated with fair accuracy before winzing. A third face justifies greater confidence, but, if provided by a winze or stope in the central and richer portion of the body, it may somewhat exaggerate the average assay value and render it necessary to apply an "assay-plan factor" in calculating the probable recovery value of the ore reserve. No such correction, however, is called for in respect of total gold, since overestimates of the assay value are offset by underestimates of tonnage, due to the fact that development does not always expose the full width of the reef. The lateral extent of the bodies is seldom sufficient to warrant more than one winze; hence blocks with four faces exposed are rare.

The correct estimation of stope width is a matter of great importance because of its bearing on tonnage and recovery value. Each block has to be carefully considered and an allowance made for the inches of hanging wall and foot wall which will probably be included in stoping. Experience has shown that the combination of a flat reef and machine mining will seldom permit of stopes less than 48 in. in height and consequently this figure is taken as a minimum, even though the reef may be abnormally thin. Experience has also shown that, regardless of the thickness of reef, at least 15 in. will be broken outside the "reef channel" and that a further allowance may have to be made because of "slips" to which the

ground will inevitably break. Taking the mine as a whole, the average stope width exceeds the average reef width by about 22 in.

The details of calculation need not be referred to, as they involve customary steps and precautions. In the end the results are compiled so as to show the numbered blocks in consecutive order, their area on dip, reef width and value, stope width and value, and tonnage. The blocks of profitable grade are then classified and cast under three headings, depending on whether they are available for stoping, in process of becoming available, or locked up in shaft or other pillars. A record is also kept of the blocks that cannot be worked to advantage in existing circumstances. It is found in practice that, on further opening, the latter frequently provide a considerable tonnage of profitable ore.

About 73 per cent. of the ore broken comes from the ore reserve, 17 per cent. from development faces, and 10 per cent. from other sources. The last-mentioned sources include selected portions of old blocks previously classified as unprofitable, new blocks not yet taken into account, pillars, hanging-wall leaders subsequently discovered, and odds and ends from old workings. Ore falling under this category is usually of lower grade than that from the ore reserve. This is still more marked in respect of development, since it is not easy to eliminate the rock from the poor faces, and even the product of the good faces suffers because of the abnormal proportion of waste which has to be included in breaking. The factors just referred to, coupled with sorting practice and milling extraction, enter in some degree into all forecasts of future recovery. A typical calculation may perhaps simplify the matter. Assume:

	Per Cent.		Dwt. per Ton
	73	From ore reserve	6.7
	17	From development	2.8
	10	From other sources	5.6
	100		
	14.5	to sorting plant excluded by sorting	5.98
Deduct			
	85.5	to mill	6.93
Deliver			
Recovery, with 96 per cent. extraction, say,			6.65

Clearly, any variations in the assumptions would correspondingly affect the result. It is necessary, therefore, in making a forecast, either to put forward a definite figure as a measure of existent conditions in the light of probability, or to give limiting figures sufficiently comprehensive to cover the various contingencies which may arise. Among the factors of doubtful bearing may be mentioned the uncertainty as to native labor, since variation in the supply may alter the relative quantity and quality of ore from the three sources mentioned, and possibly the percentage of

sorting. The position and outlook as to development may also enter. Furthermore, there is always a chance that the grade may vary more than anticipated owing to an unusual combination of the irregularities of enrichment. Of course, the number and the widely differing values of the faces afford an equalizing influence, as well as a means of adjustment, but not to the extent of insuring persistent uniformity.

In order to guard against undue depletion of the richer areas, as well as of those more favorable to cheap working, a monthly analysis of the blocks mined is prepared. This includes the fathomage and proportion taken from each block during the month, its assay value and width as determined by current sampling, and the assay value and width of the blocks as at the time of the last calculation of the ore reserve; also totals and averages. Comparison of the average assay value and width of ground stoped with corresponding figures of the total ore reserve discloses whether the ore broken was of mine grade and width. And, of course, any over- or under-mining is apparent.

Comparisons on a yearly basis are also interesting in that the monthly fluctuations are eliminated and the accuracy of ore-reserve estimates is more clearly defined. They show that as a rule the average of stope assays is slightly below that of the blocks, the winzes, as previously explained, probably exerting an undue influence on block sampling. But, as against this, they also show that the average reef width determined by stope sampling exceeds the corresponding figure for the blocks. The latter gain, as already stated, is probably due to the incomplete exposure of the reef at certain points along the levels, and the net result in practice is that the total gold recovered per unit of area closely agrees with the amount indicated by the ore-reserve estimate.

SPRINGS MINES

The remarks previously made as to the characteristics and significance of Brakpan Mines are, generally speaking, applicable to Springs Mines and, together with the specific references already made, sufficiently indicate the outstanding features of this property.

Briefly, the position is that, after six years' work and an outlay of over \$4,000,000, the mine is now ready for equipment and other expenditure that will involve \$1,500,000 to \$2,000,000 additional expenditure and fully two years' further deferment of interest. Fortunately, as already intimated, the showing of the development so far accomplished is remarkably good.

Fig. 6 will show the outlines of the company's holding, which totals 1,960 acres, as well as the disposition of the shafts and the development accomplished at the end of 1914.

The shafts are similar to those at Brakpan. They have been de-

scribed, together with some particulars of conditions and procedure, in a paper by B. D. Bushell entitled Notes on Sinking Operations at Springs

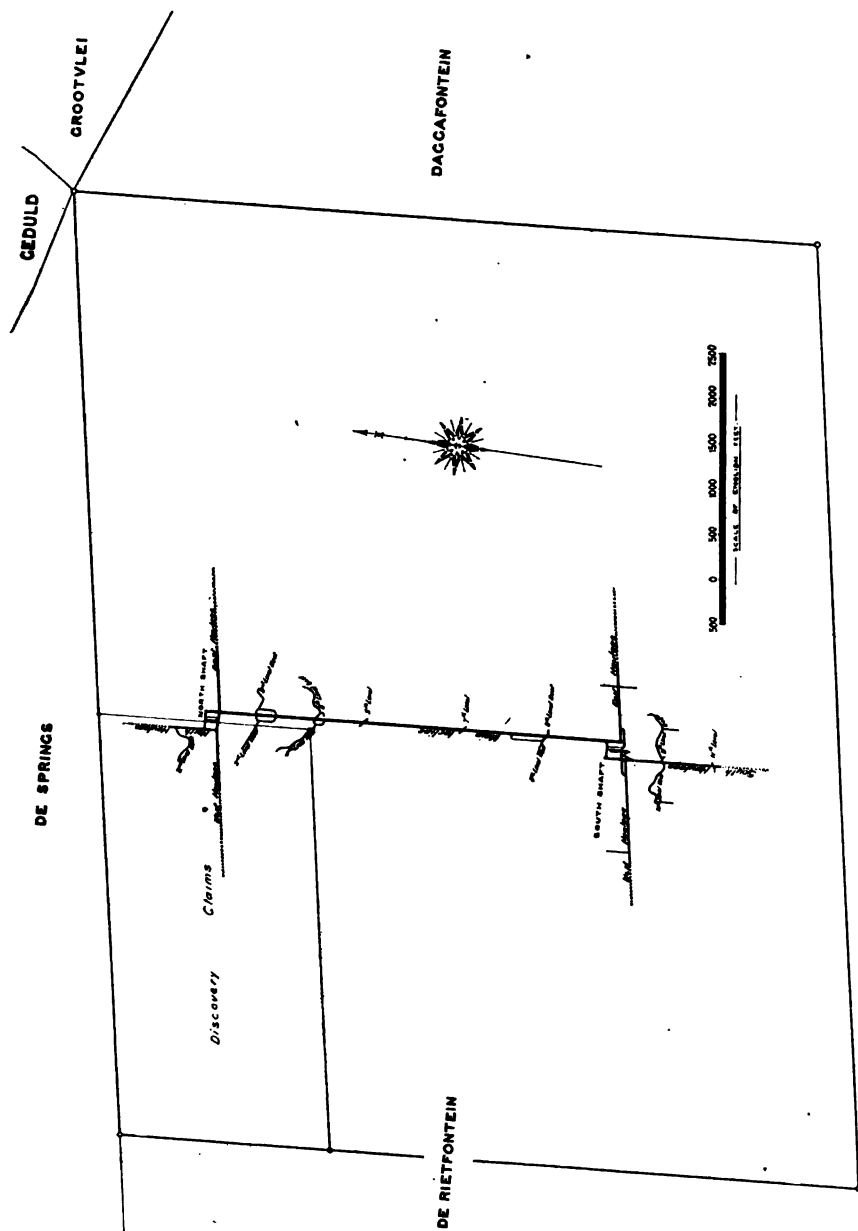


FIG. 6.—PLAN OF UNDERGROUND WORKINGS OF SPRINGS MINES.

Mines, which was discussed by the Institute of Mining and Metallurgy, London, at a meeting held on May 22, 1913. The outstanding feature of

interest was the quantity of water dealt with in passing through the dolomite. In the north shaft it was first encountered at 460 ft., and it could not be "caught up" until after the Witwatersrand quartzite was reached at 605 ft. For a period of seven months the average rate of sinking was only 27.3 ft. per month and the average quantity of water hoisted was over 2,000,000 gal. per day.

Haulageways are necessarily an important feature of the underground layout. It will be noted that six are already well advanced, and now that the main incline has been connected it is the intention to start two others from its middle point as soon as possible. In view of the Brakpan

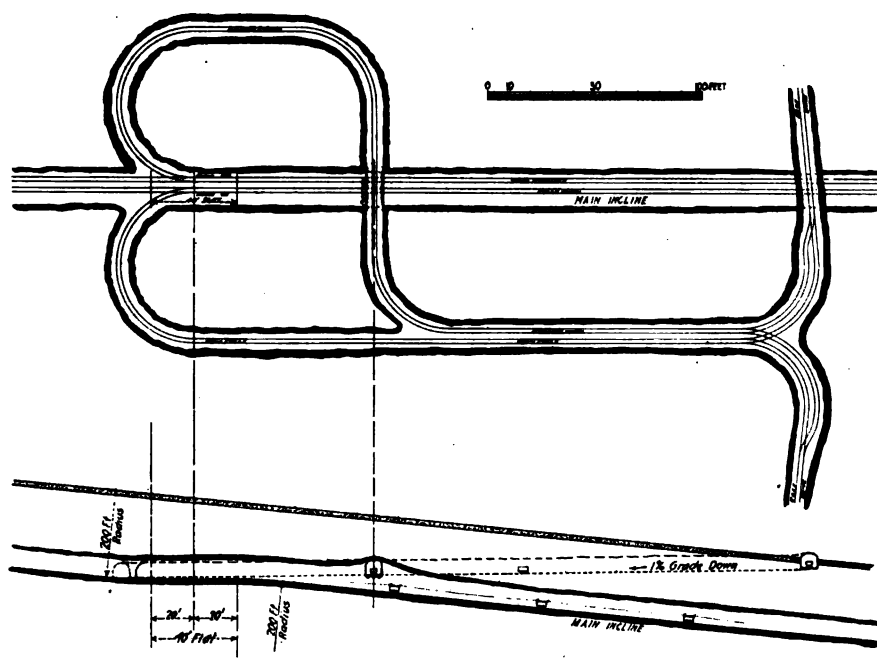


FIG. 8.—8TH LEVEL STATION OF SPRINGS MINES.

experience it was felt that the flatter dip of the reef called for still greater recourse to straight-line development. Fortunately but little water has been encountered, notwithstanding the unusual quantity in the dolomite above and the fact that faults and dikes often occur. Progress, therefore, has as yet been unimpeded and, although water may still be met with, the haulageways are now so far advanced that serious inconvenience is improbable. It is because of the absence of water and the need of economy that the sump shown in Fig. 7 is still so small.

Further reference to Springs seems uncalled for at present, Figs. 6, 7, and 8 sufficiently indicating the principal departures from Brakpan procedure.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Occurrences of Petroleum in Eastern Mexico as Contrasted with Those in Texas and Louisiana

BY E. T. DUMBLE, HOUSTON, TEX.

(San Francisco Meeting, September, 1915)

THE history of the several petroleum deposits of Texas and Mexico, or of the sediments in which they now occur, if the deposits are not indigenous to such sediments, is known in a general way only, but it nevertheless casts strong light on the conditions surrounding them and is of great importance in the search for and exploitation of them.

The petroleum deposits of Texas are separable into two main divisions:

1. The deposits of northwestern Texas, which are directly connected with the Oklahoma fields to the north, and have no representatives at all in the Mexican area.

2. The deposits of the coastal area, which extend into Louisiana on the east and connect with those of the Mexican coast to the south.

Early in the Pennsylvanian period a series of bituminous shales and limestones were laid down over a broad area in Texas, Arkansas, and Oklahoma. These are known as the Bend series in Texas, the Caney shales in Oklahoma, and the Fayetteville beds in Arkansas. Following their deposition an uplift occurred, forming the Lampasas geanticlinal, which runs northeastward from the old Paleozoic land area of the Llano region toward Red River.

The Bend series is petroliferous in places and has furnished a few producing wells on each side of the Lampasas uplift. This uplift forms the barrier and dividing line between the two divisions of the Texas petroleum deposits.

To the west of it, in waters which were the southern extension of the interior continental seas, in connection with the sediments and coal beds of the Pennsylvanian and Permian, great quantities of petroleum were deposited either as oil or as "potential petroleum." These sediments have been subject to very slight disturbance, and only very gentle folds are found, with no signs of volcanic action in any part of the area. The conditions, therefore, indicate that the greater part, if not all, of the oil in this area is indigenous to the beds in which it is found.

East of the anticlinal the oldest beds which appear in contact with the Bend are the Trinity Sands of the Lower Cretaceous. No evidence has

been found either on the surface or in wells of the presence of sediments referable either to the Coal Measures, Permian, Trias, or Jura.

To the southward, in Central Mexico, very complete sections are found of both Trias and Jura, but if the waters of those periods ever reached the Texas coast, no evidence now remains to prove it.

During the Cretaceous, however, the southern seas did cover a part of the Texas area, and the sediments of its earlier period (the Lower Cretaceous or Comanche) in the two areas are closely related. They consist of sands, clays, and limestones, which in the coastal area present little, if any, evidence of being oil bearing.

The beginning of the Upper Cretaceous deposition marked a very decided change in this respect and ushered in a period in which the formation and deposition of petroleum or petrolic matter was widespread through the coastal area.

Böse,¹ describing the section between San Luis Potosi and Tampico, mentions the limestones of the Tamasopa Gorge and their exposures in Micos Canyon, saying of them that they are probably Cenomanian or Vraconian. They form the top of the Middle Cretaceous deposits of the Mexican classification and are overlain by shales and marls of their Upper Cretaceous.

By far the greater portion of all the oil so far found in the Tampico-Tuxpam area is contained in or derived from this Tamasopa limestone.

During the period in which this Tamasopa limestone was being deposited in the Mexican region, further north in east Texas and Louisiana shallower waters prevailed and in them the sands and clays of the Timber Creek beds or Woodbine sands and clays were laid down. The Cenomanian age of these beds is indicated by Stanton,² who states that this fauna "contains species of *Acanthoceras* and other types resembling those that are characteristic of the Cenomanian in Europe."

Böse³ also correlates these Lower Cross Timber sands with the uppermost portion of his Middle Cretaceous.

These Woodbine sands are the principal source of the extensive oil production of the Caddo and other fields of western Louisiana.

Between this great Louisiana field on the north and the greater Mexican field on the south there is an interval of more than 600 miles in which these formations are not now found within the coastal area, unless some portion of the basal Eagle Ford shale may represent a time equivalent, and even if that be the case, no oil deposits are found.

Immediately overlying the Tamasopa limestone there is a series of

¹ *Guide Book, International Geological Congress*, 1906.

² A Comparative Study of the Lower Cretaceous Formations and Faunas of the United States, *Journal of Geology*, vol. vi, No. 6, p. 606 (Sept.-Oct., 1907).

³ *Neue Beiträge zur Kenntniss der Mex. Kreide*, p. 10 (1910).

limestones and clays known as the San Juan or San Felipe series and these are overlain by the Papagallos or Mendez shales.

The San Juan limestones have yielded a few imperfect ammonites, which may belong to the Taylor marl or the Austin chalk.

No fossils have ever been found in the Papagallos shales, which constitute the latest Cretaceous deposition in the Mexican coastal area and a facies of it not represented in Texas.

This divergence in geologic history began during the period of the Austin chalk with the appearance of a barrier south of Eagle Pass in the vicinity of the Sabinas River. This barrier was extended southeastward to the Gulf coast near Tordo Bay, so that in place of a continuous strand line, as at present, there were for a time two separate basins, the southern one being directly connected with the waters of the Pacific Ocean. This condition continued until the end of the Eocene; and the shales of that age found overlying the Papagallos shales south of the Tamaulipas barrier carry fossils of the California types, rather than of the Texas types such as occur in the region north of it. These shales were followed by other shales, yellow clays, sands, and limestones which carry marine invertebrate fossils throughout, those of the lower portion being of Oligocene age, while those of the upper beds are Miocene (?). It is evident from the similarity of the deposits that the conditions of deposition varied very little in this area after the close of the Eocene, when the waters of the Gulf succeeded those of the Pacific.

During this Upper Cretaceous and Tertiary time the conditions existing along the Gulf coast north of the Tamaulipas barrier were entirely different. Marine deposits alternate with those of palustrine and floodplain origin, and with periods of dry lands and æolian deposits, showing here continued variation instead of the uniformity of the Mexican area.

The history of oil deposition in the two areas is as different as that of their geology. With the close of the Cenomanian, the formation and deposition of petroleum seems to have greatly diminished in the Mexican region south of the Tamaulipas barrier, and few, if any, workable deposits of indigenous oil have yet been found in the beds of the Upper Cretaceous, or of the overlying Tertiary.

North of the Soto la Marina, since the beds are in large part direct continuations of those in Texas, there may be a similarity in such oil deposits as exist; but drilling in this area has not yet proceeded far enough to permit any definite statement to be made regarding them.

In Texas and Louisiana, however, the processes of oil formation were active at various times and pools of oil or gas are found at several horizons of the Cretaceous between the Woodbine sands and the Eocene, as well as in different members of the Tertiary. While the oil in some of them may not be indigenous, in many of them it seems to be so.

Eastern Mexico

Tuxpam-Tampico Field.—The Tamasopa limestone as exposed in Micos Canyon is composed of massive beds of hard limestone with abundant fossils. In drilling wells nearer the coast we find that while the beds of this series are still massive, they include strata in which the lime is more or less arenaceous and beds of sand, suggesting a coast line to the eastward during the Middle Cretaceous. The oscillations and land conditions of the beds of the Coastal Plain here included in the Middle Cretaceous are further shown by the occurrence in them of beds of gypsum sand of considerable thickness.

The San Juan series, which is largely thin bedded limestones in the type locality, is represented in this area by shales and shelly limestone beds with occasional heavier beds of limestone as well as of sand. They are characterized by the brown color of part of the shales and limestones and have a thickness of 400 to 800 ft.

The Papagallos shales are usually blue or gray in color, but occasionally have red bands, and the variable thicknesses shown in different wells seem to indicate a considerable erosion of these beds prior to the deposition of the overlying Eocene shales. The total thickness in this area of the Papagallos is estimated at 700 to 1,200 ft.

The Alazan shales are similar in character to the Papagallos, and it is difficult to distinguish them apart, either on the surface or in the wells, unless fossils are found in them; and, so far as we can judge by drilling, the weathered yellow surface clays in the upper members of the section in the central portion of the area are derived from similar gray marls, shales, and shells of lime. In the southeastern portion of the territory, however, somewhat different conditions of deposition seem to have existed, as was shown by a well drilled near Tuxpam to the depth of 3,787 ft. In this well the upper 500 ft. was composed of sandy materials corresponding to the Tuxpam beds, but the balance of the well was shales and gumbos. Nothing was found which could be identified with the San Juan.

The rocks of the coastal area have a general easterly dip of low angle. Near the western border, along the foot of the Cordilleras, they are strongly folded and faulted, but these folds and faults gradually diminish toward the coast. Intrusive rocks, chiefly of a basaltic nature, occur with these sedimentaries as laccoliths and sills, and in many places have reached the surface and now appear as hills and necks.

The most valuable oil pools are found along the anticlinals or domes or in faulted areas, of which the anticlinals may be the continuations. While strong seepages may occur in the vicinity of some of the volcanic necks, representing the residuum of oil which has reached the surface through fissures caused by the eruptive forces, the best wells have usually

been found away from such pronounced disturbances. The basalt occurring in the producing oil fields is mostly in the form of sills or intercalated sheets of moderate thickness.

The principal deposit of oil is that of the Tamasopa limestone. The producing pools found in the San Juan series and overlying shales are largely, if not entirely, migratory oil from the Tamasopa, which has risen to its present location through faults or fissures, and the strongest flows are apparently secured along such line of fault or fissure. There are small deposits of indigenous oil at various horizons in these shales, both of the Cretaceous and the Tertiary, but there is nothing that would indicate the occurrence in them of such oil in workable quantities.

In the Tuxpam-Tampico field we have, therefore, a vast body of indigenous oil in the Tamasopa limestone, overlain by dense beds of marls and shales which have been folded and faulted by orogenic action, permitting the collection and concentration of the oil in domes and anticlinals and along the fissures and fractured areas of the limestones and, through these, in the porous beds of the overlying shales. The intrusive and extrusive material accompanying these orogenic movements is basalt. While the movements apparently began during the Upper Eocene, the greater portion of the basalt extrusives are seemingly of late Tertiary or Quaternary age.

North of the Tamaulipas barrier there are two probable oil fields: The San Jose de las Rusias, which is a continuation of the Tampico field; and the Northern or Soldadito, which is a continuation of the Texas Coastal field.

San Jose de las Rusias.—The surface of the region lying east of the Tamaulipas Range and south of the Soto la Marina River is occupied principally by Oligocene deposits known as the San Fernando beds, through which the eruptives of the San Jose de las Rusias Range have forced their way. In this area there are seepages of oil in such quantity as to warrant careful drilling; and since beds of the Eocene seem to be wanting and the San Fernando beds occur in direct contact with the Papagallos shales, the Tamasopa limestone should lie within reach of the drill.

Soldadito.—The coastal strip lying between the Rio Grande and the Soto la Marina is occupied by deposits which are direct continuations of those north of the Rio Grande; and similar indications of oil, sulphur, and gypsum are present at certain localities. The only well of which we have record, although it did not exceed 750 ft. in depth, showed both sulphur and oil.

Northwestern Texas

In this field we have an example of the occurrence of indigenous oils which is of interest from the fact that the beds containing them, compris-

ing the entire series of deposits of the Carboniferous and Permian periods in this area, show comparatively slight effects of earth movements and absolutely no traces of vulcanism. The petroleum deposits, therefore, having been subjected only to minor extraneous causes of change, are as nearly as possible in the condition of their original deposition or formation. Low domes and low broad anticlinal or monoclinal folds furnish the collecting places for the commercially valuable pools; and the surface indication of these is often so slight as to require instrumental surveys to determine their presence. Oil pools are found in sands of the Wichita, Cisco, Canyon, Strawn, and Bend series. The character and occurrence of the oil sands are well described by Udden:

"The position of the sands in the entire section suggests that they are ancient sand bars and perhaps beach sands, built up, washed away, rebuilt, and finally buried under accumulating argillaceous sediments, during a long period of more or less gently changing geographic conditions, involving, on the whole, a progressive sinking of the shore-land and the adjacent bottom of the sea. Sands connected in one place may in another place be separated. Closely contiguous sands may be wholly separate. Sands clearly interrupted at some point may be connected by some devious circuit in an unknown direction."⁴

This field brings out strikingly a relationship between coal beds and oil deposits, which, if it exist elsewhere, is not so apparent.

The Cisco division of Cummins carries Coal Seams 7 and 8 of his section. These are found outcropping from Bowie eastward through Jack and Young Counties. Near the Colorado River in Coleman County, these seams have been opened up and are known there as the Chaffin and Bull Creek seams; and the exact position of these coal beds as regards fossiliferous horizons of this series is well shown by Drake's report⁵ on the Colorado Coal Field of Texas.

A well drilled west of Henrietta on the Halsell farm to a depth of 3,985 ft. furnished a series of samples and fossils which were studied by Udden. The results⁶ show that the well penetrated the entire Coal Measures series of the Pennsylvanian and entered the underlying Bend. The relationship of the oil and coal horizons is thus described:

"It appears probable that the horizon of the Chaffin coal in the Cisco division corresponds to the dark blue shale reported as underlying the gray lime at from 1644 to 1655 feet below the surface in the Halsell boring, and that the Bull Creek coal is the stratigraphic equivalent of what is reported here as dark blue slate lying from 35 to 70 feet below the 'gray lime' at from 1420 to 1436 feet below the surface. It will be recalled that Drake reports having observed *Fusulina cylindrica* about 150 feet above the Bull creek coal in considerable abundance,⁷ and that this is the uppermost part in

⁴ Bulletin No. 23, University of Texas, p. 100.

⁵ Fourth Annual Report, Geological Survey of Texas, pp. 357 to 446 (1892).

⁶ Udden: Bulletin No. 23, University of Texas, pp. 82 to 83.

⁷ Report on the Colorado Coal Field of Texas, Drake and Thompson, Fourth Annual Report, Geological Survey of Texas, p. 413 (1892).

the Colorado section from which he reports this fossil. Likewise the limestone at 1445 feet below the surface in the Halsell well is found to contain this fossil, and no rock higher up in this well seems to be of a kind in which this fossil is at all likely to occur, excepting the other thin limestone reported at the depth from 1420 to 1436 feet. This part of the Halsell well section is doubtless also the equivalent of the deeper productive oil and gas sands in the two fields under investigation. These consist of shales, limestones and sandstones, which lie at from 1500 to 1700 feet below the surface in the wells near Petrolia and at from 1800 to 2000 feet below the surface in the Electra field. This general correlation seems to be warranted by palaeontologic evidence as well as by evidence based on the lithologic character of the beds explored by drilling."

This well is probably 30 miles or more from the outcrop of the coal; and we cannot now say just what part of that distance is underlain by the extension of the coal bed before its replacement with the oil shales and sands, or how near the coal and oil deposits are to each other.

Coal seam No. 1, in the Strawn division of Cummins's section, has been worked for many years near Thurber and Strawn; and recent drilling has found oil a few miles west of the mines. The well records here, as studied by Kennedy, show that this coal maintains its thickness for some 3 or 4 miles west of the present workings, and that it then gradually thins to the west, is finally missing entirely, and oil is found in the same horizon in sands and shales. The distance between known coal and known oil is only about 500 ft. Furthermore, a well $2\frac{1}{2}$ miles farther west shows bituminous shales occupying this horizon; and neither coal nor oil was found in them.

The conditions here seem best explained by Murray Stuart's idea of the sedimentary deposition of oil. There was seemingly a coal swamp or lagoon along the sea coast, the petrolic decomposition of the organic débris of which furnished the oil which was carried out and deposited in the contemporaneous sediments of the sea floor.

The Coastal Area

Cretaceous.—While some impregnations of asphalt occur in Burnett County in connection with certain beds of the Trinity sands, these are probably remnants of migratory oil from the underlying Bend series. Similar migratory oil occurs in the same sands in Montague County. No other occurrences are reported from beds of the Lower Cretaceous.

In the typical section of the Upper Cretaceous of central Texas the Timber Creek or Woodbine beds were followed by the Eagle Ford, Austin, Taylor, and Webberville beds. In eastern Texas and Louisiana the lithologic character of these beds is somewhat altered, and other names are used to designate their representatives.

The active deposition of Cretaceous oil began with the Woodbine series. These sands outcrop from Woodbine south of Red River to the Brazos near Waco; but in this region, if they were ever petroliferous, the

oil has been forced out by the inflow of water, and now they are water-bearing beds, and no signs of petrolic deposits are known in them.

South of the Brazos the Woodbine is overlain by the Eagle Ford and Austin and does not appear at the surface. Shallow wells on the South Bosque River west of Waco, drilled in 1890 by Col. William L. Prather, found small quantities of light oil in the Woodbine. This field was developed in a small way later, and is the only occurrence from these sands so far recorded in central Texas.

From Woodbine the outcrop stretches eastward into Louisiana, and in the Caddo region the sands are highly petroliferous and furnish the best wells of that belt. The productive oil and gas of the Eagle Ford is confined to the "Blossom sand," which is unknown as such on the Colorado. Gas is plentiful in the Blossom sand in the Caddo field, but such oil as is found there is heavy, and producing wells are rare if not unknown. The bituminous and even petroliferous character of the shale is apparent, however, at many places along its outcrop in central Texas, and it furnishes seepages of a heavy tarry oil from small wells near Waters Park, north of Austin.

The Annona chalk, which is an eastern representative of a portion of the upper Austin or basal Taylor, is a good oil horizon, and the Nacatoch sand, probably the equivalent of part of the Taylor, is an excellent gas horizon. Thus, in Louisiana, there are four distinct horizons of gas and oil in the Upper Cretaceous, the principal of which is the Woodbine, and it is probable that the oil and gas in these beds are largely indigenous.

The deposits of commercial value occur in connection with folds and domes, and at present are best developed in the area between the Sabine and Red Rivers. Harris's idea is that the great accumulation of oil in this area is due to an extensive crustal movement, which he calls the Sabine uplift, and which has brought the Upper Cretaceous beds to within 700 ft. of the surface over a large area.⁸

West of San Antonio a few wells drilled in Medina County developed small quantities of oil in the Austin chalk; and it is probable that later these may prove to be of value.

Near Cline, west of Uvalde, there is a deposit of asphaltum which occurs in the Anacacho limestone. The Anacacho of this area may be the equivalent of the Annona of the East Texas section.

The other occurrences west of the Sabine River are confined to the Taylor beds. The pools at Powell, Corsicana, San Antonio, Thrall and other localities find their supply in these beds. In some of these the oil is possibly indigenous, but at Thrall it is probably migratory.

Toward the end of the Austin there was an outbreak of vulcanism in this area and active volcanoes were in operation around Austin. Pilot

⁸ Oil and Gas in Louisiana, *Bulletin No. 429, U. S. Geological Survey*, p. 27 (1910.)

Knob and other necks are evidence of their character and the mingling of the volcanic ash with the sediments of upper Austin and lower Taylor gives the date of their activity.

From drilling so far done at Thrall it would appear that just after the close of the Austin there was at this place a flow of vesicular lava (probably on the floor of the sea) which was later covered by the sediments of the Taylor. It is in connection with this eruptive material that the Thrall oil occurs. The fact that this eruptive action took place, and the similarity of the oil to that of the Woodbine, suggest the possibility that the oil may have originated in the Woodbine sands and reached its present location through rifts and fissures made by volcanic action in the firm sediments of the Eagle Ford and Austin.

At present we have no direct evidence bearing on the duration of activity of these volcanoes, but there is nothing in the surroundings of the old necks, so far as known, to suggest that they were centers of eruption throughout any extended period. The Pilot Knob volcano, however, marks the beginning of a series of earth movements that continued at intervals well into the Pleistocene. The effects of the earliest known succeeding movement are found in the numerous Cretaceous domes and ridges of eastern Texas and Louisiana, which are, seemingly, of pre-Tertiary age, since the domes include beds of Webberville age and are surrounded unconformably by the sediments of the lower Eocene.

A later movement caused the Sabine uplift, which involves not only Cretaceous beds but lower Eocene as well.

In connection with the Jackson and Oligocene of eastern Texas, we find several extensive deposits of volcanic ash, which shows the existence of volcanoes or submarine fissure eruptions near enough to permit such accumulation; but it can hardly be supposed that these were derived from the known volcanoes of the Cretaceous.

Eocene.—The deposits of Eocene age, in spite of their wide areal distribution in the Gulf Coastal Plain of Texas and Louisiana, have yielded comparatively small amounts of petroleum.

No oil whatever is known in the Midway or Wilcox of the Lower Eocene and only two small pools have been found in the Marine beds which overlie them. At Oil Center, Nacogdoches County, a small body of oil of a lubricating grade was found, and another small amount of similar oil was found in these beds at Crowther in McMullen County, some 300 miles southwest of the first. While oil indications may occur at other points, no workable deposits are known.

The Yegua sub-stage of the Claiborne, however, has proved to be a very valuable gas horizon in the region between the Sabine and the Rio Grande; and we confidently expect that it will equal in productiveness some of the sands of the Cretaceous, or even those of the Carboniferous, when it shall have been properly exploited.

Beds of the Jackson and Oligocene, which followed these, are equally destitute of oil.

Harris found a few Jackson fossils in two wells at Sour Lake and a few others were found in a well at Saratoga. The relations of these beds to those of the surrounding Neocene, however, show that their presence is due to folding or faulting; and it is probable that any oil they may carry is migratory, since the well at Saratoga penetrated the Jackson sediments for a hundred feet and found no indications of oil in them, but instead, a sand carrying a very large volume of sulphurous water of high temperature. The only oil found in this well came from a series of blue shales, thin beds of marly limestones and sands, about 200 ft. above the top of the limestone carrying Jackson fossils. It may be remarked here that although a number of wells were drilled in the same vicinity to depths over 200 ft. greater than the depth at which the Jackson limestone was found, these wells all ended in the same blue shaly clays and marly limestones without any appearance of this heavy fossiliferous limestone. At Sour Lake the same conditions exist. Jackson fossils were found in a well at the depth of about 1,500 ft., but no oil was found in it. Wells drilled to much greater depths than 1,500 ft. within the territory lying between Saratoga and Sour Lake do not show beds of the Jackson stage.

The probabilities are, therefore, that while the conditions were favorable for the formation and deposition of vast quantities of lignites and an abundance of other organic matter during the Eocene, the conditions for the formation or storage of petroleum were comparatively unfavorable.

Neocene.—The oil of the Gulf Coast proper all occurs in sediments of Neocene age, often in or around domes, and frequently in connection with deposits of salt, gypsum, and sulphur.

The Neocene section was first differentiated on the Nueces River in southwest Texas, where the basal sands and clays carrying, in their higher beds, fossils of Loup Fork or Upper Miocene age were called the Oakville. The sands overlying these, with Pliocene fossils of Blanco age, were called the Lapara, and succeeding these were the Lagarto clays and the Reynosa. Neither the structure nor the character of these Oakville or Lapara beds lends itself to the formation or retention of petroleum. In the south central extension of the country underlain by them several wells have been drilled completely through these deposits without showing any signs of either gas or oil. Taking them as a whole, they are non-bituminous and may be looked upon as dry.

East of the Brazos this exact lithologic succession has not been found, and in east Texas the term Fleming clays has been used for a series of deposits which, as exposed on the surface, are of palustrine and fluvial origin, and, as evidenced by the fossils collected from them, may represent the entire Neocene section below the Lafayette or Reynosa.

The shore deposits of the Fleming, as shown in surface exposures, are

for the most part blue and greenish clays carrying great quantities of calcareous nodules, calcareous cemented sandstones, thin lenticular nodules of brownish to brownish yellow ferruginous sandstones, and occasional heavy deposits of the same material, and of arenaceous limestones. The vertebrate remains are mostly found in the blocks or large boulders, but sometimes appear in the clays.

The seaward extensions of these deposits are very different from the shore deposits. Well records show below the sands, gravels and clays of the Lafayette:

1. A series of brown shales or shaly clays with streaks of gumbo. These have a thickness of 300 to 500 ft. and possibly represent the Woodville.

2. Thin to heavy bedded interstratified shales, sands, and gumbo, with shells of rock and some gypsum, underlain by massive beds of gumbo, blue and brown shale, gravel, and sand, probably representing the Burkeville, 1,000 to 1,500 ft.

3. Interbedded shales, sands, limestones, and gumbos similar in character to those of the upper portion of the preceding series, but occasionally of more massive bedding. These probably represent the Navasota-Oakville Miocene horizon, and have a thickness of more than 1,000 ft. Some portions of these beds, as shown by fossils found in them, are of marine origin, but the exact relations of the marine and palustrine deposits have not been ascertained.

These beds thicken considerably toward the coast and complete sections are only found in undisturbed areas. In the vicinity of domes or folds some portions, if not all, of these beds are usually lacking, such as may have been deposited having been removed by erosion.

These beds are fossiliferous, but unfortunately the pieces of shell coming up in the drilling are usually very small and unidentifiable, although at some points fairly large pieces of small oysters are obtained. At Saratoga, however, shells are found in the shales at 1,200 ft. which are identical with those of Galveston Deep-Well Miocene at 2,400 ft.; and similar shells are also found in wells at Batson.

The shore deposits of the Fleming carry a fair-sized fauna. In the region around Alexandria in Louisiana the fauna is altogether of brackish-water forms, but at Burkeville we have a mingling of brackish-water forms with vertebrates; and at Navasota, where the deposits are in a great measure calcareous clays and limestones, they carry vertebrate remains and rolled Cretaceous shells. Some of the deeper-water shales and clays carry the Burkeville brackish-water fauna, and it is found at Pine Prairie at a depth of 1,545 ft., at Anse la Butte at 1,550 ft., at Edgerly at 3,000 ft., and at Terry at 3,100 to 3,300 ft.

Many of the invertebrates found at Burkeville and eastward are new, and there is little that is useful for exact determination of age. Dr.

Dall considered them Pliocene.⁹ No other determinable invertebrates were found in the series.

The vertebrates were studied by Dr. Matthew, who concluded that they embraced species from Middle and Upper Miocene and possibly Lower Pliocene.¹⁰ The older beds occurring in the Trinity and Brazos drainage carry invertebrates of Upper and Middle Miocene age, and, therefore, probably represent the Oakville of west Texas. Teeth of *Protohippus* or long-crowned *Merychippus* were found in the same beds with the Pliocene shells near the base of the section at Burkeville; and it is possible that this portion of the beds is Pliocene and corresponds in age to the Lapara of west Texas, while the higher horizon at Woodville may represent the Lagarto, which it closely resembles lithologically.

While the shore deposits of the Fleming beds are non-bituminous and no sign of either oil or gas has ever been found in them, the seaward extensions are bituminous to a considerable degree. Small pieces of lignite and asphaltic material have been reported from a widely extended series of wells. Shows of oil appear in these beds at various horizons and good wells have been obtained in them. Wells in the Saratoga field drilled to a depth of over 1,200 ft. obtained their oil from these shales, and the wells at Terry get their oil from the Burkeville horizon. The large producers along the eastern side of Humble and the western side of Sour Lake fields appear to obtain their supplies from sandy shales belonging to this series.

The Lafayette, which closes the Pliocene in this area, shows in some wells a thickness of 500 ft. of sands, gumbos, and clays. They carry water in abundance, but no oil.

Of the overlying Pleistocene deposits the Beaumont clay, of Port Hudson age, is most characteristic. These clays and sands are very variable in thickness. In places the Pliocene beds are found in low hills, surrounded but not covered by the clays, while in others the Beaumont clays show a thickness of 2,500 ft. and more, above these beds.

The probabilities are, as stated, that all the oil of the Louisiana and Texas coastal belt is of Neocene age. It is separable into two classes, shale oil and dome oil. The first is regarded as indigenous to the beds in which it is found, and the second as migratory oil derived from it.

The relations of these two classes of oil are particularly well shown in the Humble field.

The dome oil was first discovered, and the field brought in, in 1904. Development and comparative exhaustion were very rapid, the climax of production, 90,000 bbl. per day, being reached in June, 1905, after which it declined to 20,000 bbl. by December of the same year. The

⁹ *Water Supply Paper No. 335, U. S. Geological Survey, p. 73 (1914).*

¹⁰ Dr. W. S. Matthew, in literature.

underground conditions were the same as those of other dome fields; the wells were comparatively shallow, being usually less than 1,400 ft., and the oil, like that of the other dome fields, was of 20 to 24 gravity B $\acute{e}$. The plug of underlying salt was found at 1,400 to 1,600 ft. over an extensive area.

As the production of this central pool declined, wells were sunk at various distances from it; and the Esperson wells, a mile south of the dome, found a pool of light oil in shales. Later, similar oil was found in shales north of the old field, and more recently large producers have been secured in the shales from 1 to 2 miles east. This oil has a gravity of over 30° B $\acute{e}$.

As shown by the well records, these shales are found on the west, north, and eastern sides lying well up on the gypsum and other mound-forming material, but not surmounting the same.

On the western side, a well about 1½ miles northwest of the main portion of the field or dome was drilled to a depth of 3,015 ft., of which the last 1,000 ft. were gumbos, rock, and shales, with some sand and streaks of gypsum. These beds evidently belong to the series of shales, gumbos, and rocks we are now considering. At the depth of 2,200 ft., a small show of oil was found, and small quantities of gas were met with at various depths down to 2,800 ft.

On the south side of the San Jacinto River and north of the original field, a number of wells have been drilled to depths varying from 2,700 to 2,900 ft. These wells have been small and irregular producers, obtaining their supplies from what has been termed a broken sandy shale, or thin sands interstratified with thinly bedded or laminated shales.

Along the eastern and southeastern sides of the Humble field wells have been drilled to depths exceeding 3,300 ft. In a well on the Stephenson Survey the gumbo, rock, and shale beds appear to begin at about 2,400 ft. These are the producing beds from which all the outlying wells on the north, northeast, east, and southeastern sides of the Humble field obtain their supplies.

How close to the central portion of the Humble dome these shales approach is not known, but in the Crosbie well on Block 12 of the Landslide the same shales occur at a depth of 1,500 ft. and in the Esperson wells on the Hermann tract in the Jones League at a depth of 1,650 ft.

These shales appear to thin out toward the dome. Thus, in the deep wells on the southeastern and eastern sides they appear to have a thickness of at least 1,000 ft., while in the Esperson and Crosbie wells nearer the dome they have thinned to 700 ft., and disappear altogether in the Landslide or Cherry tracts before reaching the Echols ridge or mound proper. Along the north and western sides the same thinning and disappearance are found.

Similar conditions exist in the Sour Lake field, at Spindletop, and elsewhere.

From the condition in which these shales lie and their relation to the mounds it is clear that the salt and gypsum and other beds forming the mound-making materials are much younger than the shales. Not only does the general tilting show this, but it may be observed that at many places the salt itself has been found intruding into the shales. This is the case at Anse la Butte, Vinton, South Dayton, and Hoskins Mound.

The origin and manner of formation of the domes of east Texas with their great bodies of salt, gypsum, and sulphur has been the subject of much discussion, but without an entirely satisfactory explanation being reached.¹¹

Whatever the causes back of them may be, they became operative in late Cretaceous time, were active in the Eocene, and again in the Neocene and Quaternary, as is proven by wells drilled in and around the domes of the different belts. From the evidence it appears quite certain that the Sun mounds near Waller and Damon's Mound are post-Lafayette.

The very variable thickness of the Port Hudson, as already noted, evidences the extremely irregular surface of the Pliocene deposits on which the Beaumont clays were laid down, a condition which is in great contrast with the present level surface of this area.

Possibly nothing could bring out the differential movements that have taken place in this coastal area since the Miocene deposition more clearly than the fact that while the Pliocene brackish-water fauna which occurs at Burkeville 150 ft. above sea level is found in abundance at Terry, 66 miles south, at a depth of from 3,000 to 4,000 ft., the marine Miocene fauna which occurs at Saratoga at a depth of 1,000 ft. is only 2,400 ft. deep at Galveston, 74 miles south of it.

It seems clear, therefore, that during the Neocene the coastal area of east Texas was subject to extensive oscillation, and it is probable that these movements rather than those of earlier date were directly connected with the formation of the domes and folds found here.

The relation of the bodies of salt, gypsum, and sulphur of these domes to the surrounding sediments indicates that these masses have certainly penetrated 2,000 or 3,000 ft. of the sedimentary strata. The clays, sands, and limestones immediately adjacent to or overlying them are tilted at comparatively high angles for this region, the surrounding sedimentaries dip away from them at lower angles, and beds or sills of salt and gypsum extend from the main mass out into the surrounding beds. The great similarity existing between these occurrences with their bosses of salt

¹¹ A late paper on the subject which seems a step in advance of former ones is by E. G. Norton, Origin of the Louisiana and East Texas Salines, *Bulletin* No. 97, Jan., 1915, p. 93.

and gypsum and those of the intrusive basalts of the Mexican field is well worthy of attention.¹²

The origin of the oil found in the coastal area has been variously explained. Its possible derivation from beds of the Permian or Cretaceous age has been suggested under the hypothesis that the oil reached its present horizon through faults formed in connection with the uplift of the domes. These fissures must have extended through the overlying Eocene and Neocene sediments for this to have been true.

We have no evidence whatever of any Permian deposits southeast of the Lampasas geanticline nor of the continuation of the Woodbine as an oil horizon so far southward as the coast.

The lowest horizon reached in any of the oil-pool wells is the Jackson, which shows no signs of oil. Below this there is, above the Woodbine, at least 2,000 ft. of Cretaceous and 2,000 ft. of Eocene strata. These are mostly uncompacted sediments with many water-bearing beds under a heavy head. Any fissure line which would make a passage for oil from the Woodbine to the upper sands would be equally open to water from these various intermediate beds, and, judging from the results of similar conditions in well drilling, the water would soon drown the oil and close the fissure by caving.

If such transmission of oil were possible we should find some evidence of it in the Caddo region (where such fractures also occur) in the presence of oil in various porous beds above the Woodbine, but this is far from true. Such oil as is found above the Woodbine gives no suggestion of having migrated upward from these beds.

Another strong presumption against the pre-Neocene derivation of the oil is that we know of no oil in the Eocene, and the oil of the Cretaceous has a paraffin base, while all of the coastal oil has an asphalt base.

The close association of the oil, gypsum, salt, and sulphur in some of the domes has naturally suggested the idea of a common origin or a close relationship in origin, and this has been widely discussed. Were this true, the oil should be found in connection with all such domes, and such is not the case. We have oil pools where there is no dome and where no salt has been found, and we have numerous domes and bodies of salt, gypsum, and sulphur without any accumulations of oil.

A simpler solution by far, and one which seems fully justified by the facts, is that the oil of this area was formed in the usual manner, deposited in beds of Neocene age, and was collected into pools and concentrated by the movements which may also have given rise to the domes. When the uplift which caused one of the domes was in the vicinity of one of these pools it captured a part or all of its oil, and where there was no pool near it the dome had no oil.

¹² Garfias: The Effect of Igneous Intrusions on the Accumulation of Oil in Northeastern Mexico, *Journal of Geology*, vol. xx, No. 7, p. 666 (Oct.-Nov., 1912).

Conclusion

Possibly the greatest contrast between the occurrences of oil in Mexico and those in Texas and Louisiana is, that while the Mexican deposits so far as developed practically originated in a single horizon, the indigenous oils of Texas and Louisiana not only occur in great quantity at a similar horizon, but are found at many other horizons, both below and above, ranging from the Pennsylvanian to the Pliocene.

The Mexican oil occurs in a field which has been subjected to the action of strong orogenic forces, the intrusive and extrusive material associated with the movements being basalt.

The Texas and Louisiana fields embrace deposits in areas in which evidence of orogenic action is very slight, and in which no intrusive or eruptive materials are known, as in northwest Texas; those in which it has been greater, and in which the eruptive or intrusive materials are basalt, as in central Texas, and rock salt, as in western Louisiana; and finally of those which have been subjected to much greater movements, associated with which, as intrusive and protrusive materials, we find salt and gypsum. With these sulphur is connected, possibly as a secondary product.

The Mexican oil pools are apparently of greater extent and the conditions surrounding the occurrence of oil in them are far more uniform than those of the coastal oil fields of Texas and Louisiana. Consequently, the questions arising in connection with the discovery and exploitation of the oil are much simpler and easier of solution.

Taken as a whole, the Mexican field will probably show a greater aggregate area of oil pools, will furnish wells of greater volume and will be longer lived than the fields of Texas and Louisiana.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Slime Agitation and Solution Replacement Methods at the West End Mill, Tonopah, Nev.

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THIS paper deals with only one step in the treatment of ore at the West End mill; not because the other steps are repetitions of practice in other mills, but because in this particular step there is in use the Trent agitator. This device, although it has apparently failed in many mills, is here giving excellent service and has proved to be well adapted for making a thorough replacement of pregnant solution with a minimum amount of barren solution. The strong and weak points of this agitator, its present simple construction, and its use as a thickener for agitator or battery pulp are features of interest; but perhaps the point of greatest interest to the metallurgist in this day of continuous decantation is its use in a series of tanks for slime treatment by the replacement method.

The agitating department of the West End mill consists of six redwood tanks, 24 ft. in diameter, 18 ft. high, each equipped with a Trent agitator, a centrifugal pump, and a motor. The pulp is transferred by a pump from the top of a flat-bottomed tank to a set of arms and nozzles in the bottom of the tank, at just sufficient pressure to cause the arms to revolve. The streams from the many nozzles keep the bottom of the tank clean, and these streams, coupled with the effect of the revolving mechanism and the ascending current, keep the pulp in constant motion and of constant gravity.

These agitating tanks hold 90 tons of dry slime and 202 tons of solution with a 1.24 pulp, our ore having a specific gravity of 2.7. This capacity is about 10 per cent. in excess of the standard 15 by 45 ft. Pachuca tank. The tanks are connected in one series for continuous agitation, the pulp for each agitator being drawn by its pump through a branch suction from near the top of the preceding tank and delivered to the bottom of the tank by the agitator arms along with the regular pulp of that agitator. This method works automatically, since the flow through the branch suction varies directly with the difference in head of the two tanks. The chance for new pulp to pass out quickly is, therefore, much less than in many other systems of continuous agitation, since the new

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pulp is delivered at the bottom of the tank, the discharge from the tank is near the top, and the ascending current is uniform and slow.

The pulp is delivered to the first agitator of the series at a gravity of 1.26 by a set of diaphragm pumps, which raise the pulp to a height of 10 ft. above the top of the battery-pulp thickeners, whence it flows first in an open launder and then in a pipe to the suction of the first agitator pump. At intervals in the open launder are placed pieces of battery screen at an angle, which, with those placed at the lip of the diaphragm pumps, catch and remove nearly all the lime scale, wood pulp, and like material from the slow-traveling thickened pulp.

The pulp from the last agitator is raised by an air lift to a Dorr thickener, 28 ft. in diameter and 22 ft. deep, where the pulp is thickened before filtering.

The slaked lime and half the lead acetate are added direct to the scoop box of the tube mills, while the greater portion of the cyanide and the rest of the lead acetate are added at the lip launder of the diaphragm pumps, and the remainder of the cyanide is added in the third agitator. For heating the pulp, live steam is introduced into three of the agitators, Nos. 1, 3, and 5, the purpose being to keep the temperature during agitation around 110° to 120° F., in order to hasten extraction and obtain good extraction with the use of lower-strength cyanide solutions. These high temperatures are readily maintained in a Trent agitator and are not uncomfortable to the operators, since the heavy coat of foam on the agitators acts as an insulating blanket, keeping the pulp from radiating its heat or moisture, as it does in many other types of agitators. The condensed steam adds to the tonnage of mill solution, but it freshens the solution by dilution of impurities and it is needed to replace the solution that is mechanically lost from the filter.

Once a day, at 4 a.m., pulp samples are taken from agitators Nos. 1, 3, 5, and 6, which are thoroughly washed and dried and made ready for the assayer at 7 a.m. Normally, these assays show respectively 50, 25.5, 19, and 15 per cent. of the valuable content of the agitator heads.

Air for the proper aeration of the pulp is compressed to 16 lb. in a compressor of 100 cu. ft. piston displacement, and is fed into the discharge pipe of the agitator pumps. By the time the air is discharged from the nozzles it is thoroughly dispersed in the pulp as minute air bubbles. These air bubbles slowly make their way to the surface and escape into the foam. This well-distributed aeration, in contrast to the short contact of pulp with air in any air-lift type of agitator, is much in favor of the Trent. The pumps of the agitators can be made to suck their own air by throttling the main pulp suction, but in a large installation it is probably more economical to use an air compressor.

The original agitators as installed in 1911 were of the Trent underfeed type, with the large grit-proof bearing in the bottom of the tank and with

the arms fitted with 15 $\frac{3}{4}$ -in. nozzles. While these agitators gave excellent service, there was a gradual wear, which necessitated repairs on the grit-proof bearing, due to absorption of the air in the air chamber and consequent rise of slime pulp to the ball race. An occasional cleaning of the nozzles was also necessary, due to their choking with bits of wood pulp,

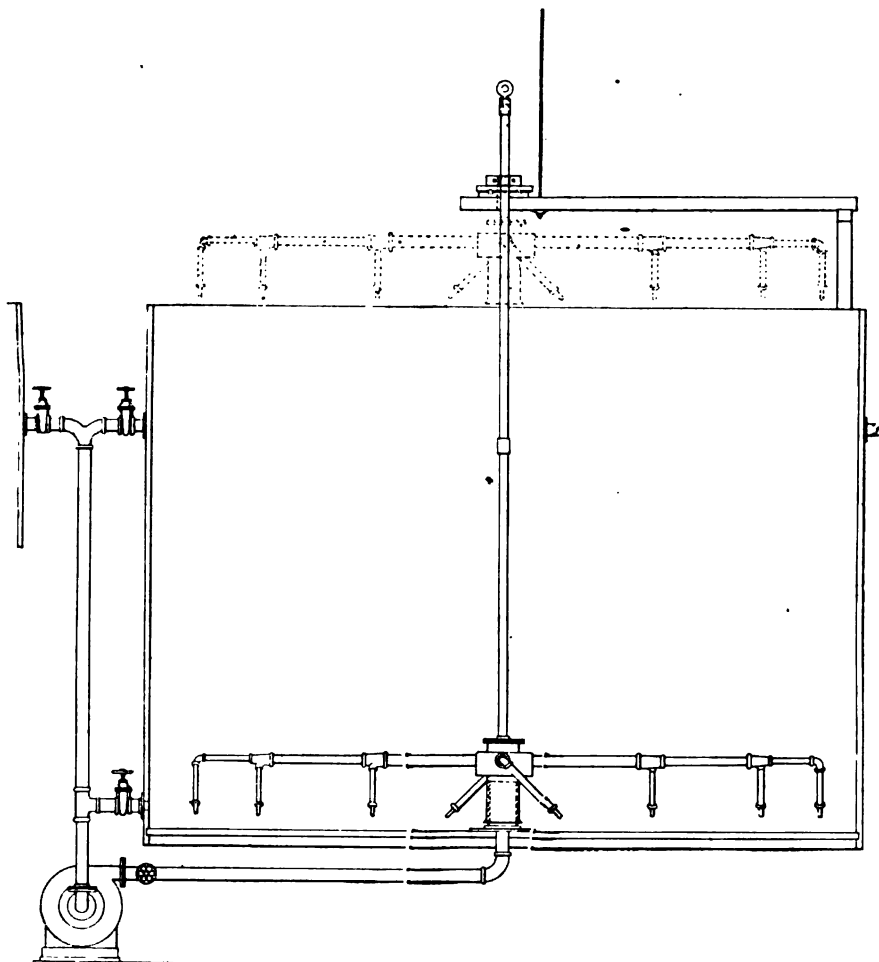


FIG. 1.—TRENT AGITATOR, CARPENTER TYPE.

waste, and chips, as a nucleus, surrounded and packed tight by sand from the pulp. On a charge system these repairs could be taken care of easily when the tank was emptied, but when the continuous system was adopted it was necessary once in two or three months to empty each agitator into the following unit by the aid of the transfer pump, and make these repairs, to avoid the danger of an agitator stopping at a time when

there was not room in the mill circuit to empty it. The author, after seeing a Trent agitator suspended from a ball race at the Tonopah Extension mill, because the lower submerged ball race was worn out, conceived the idea and built a new type of this agitator which could be raised, cleaned, and dropped back into position without disturbing the pulp level in the tank. It consisted of a ball race, on timbers over the center of the agitating tank, from which the agitator mast, built of 3-in. pipe, was suspended, a collar clamped around the upper projection of the pipe resting directly on the ball race (Fig.1). At the same time, the large upper and lower castings, weighing 800 lb., were discarded, and castings weighing 300 lb. were substituted, which were fitted with extra heavy pipe nipples to serve for the slip joint between the stationary pump discharge and the revolving arms. The upper nipple was of 8-in. pipe, 12 in. long, and was made bell-shaped at the bottom. The lower nipple was of 7-in. pipe, 10 in. long, chamfered at the top. These nipples, after a skim cut in the lathe, have $\frac{1}{64}$ -in. clearance between them and about a 9-in. contact, which allows only a small flow of pulp to pass between them, but permits the agitator to be raised 9 in. before disengaging. An eye bolt was fastened in the roof directly over the center of the tank, and by means of a chain block the agitator is readily raised above the top of the pulp, where every part of the agitator, excepting the 7-in. nipple in the base casting, can be inspected, cleaned, and repaired if necessary. Due to the balanced arms, the bell-shaped nipple, the plumbed point of support, and the guiding ball race, there is no difficulty in lowering the agitator and again engaging the nipples. The first agitator of this type was placed in commission on Oct. 20, 1913. On account of its lighter weight, it traveled faster and put very little weight on the ball race, as the upward reaction from the nozzles nearly counterbalanced the weight. It was raised, cleaned, and lowered with ease. The tank was emptied at first every couple of months to examine the nipple wear, which was found to be very small. The original pair of nipples, after 19 months' service, are still in use and the wear has not affected the speed of the agitator.

After a thorough trial this new type of agitator was placed in all the tanks at convenient times. It has never been necessary to empty an agitator tank because of failure of the agitator of this type to revolve or keep the pulp agitated. The oldest agitator of this type had operated 10 months (on May 15, 1915) and three others nine months without lowering the pulp in the tank; and, judging from the speed of rotation and the feel of the bottom of the tank, they are good for months to come. When the agitator has been in operation for from four to eight weeks a rod drawn along the tank bottom will strike ridges, which are indications of choked nozzles. After raising the agitator and cleaning out an average of a half-dozen choked nipples, the agitator is lowered to within 6 in. of its usual depth until the cleaned nozzles have cut away the ridges.

These agitators have been shut down for hours at a time and have started off after a few minutes' aid in making the first revolution.

This new type of agitator has now been adopted by the Trent Co. as its standard design, adding to the previous good points of this agitator those of simplicity, durability, and reliability.

The pump used on the agitator is a 4-in. centrifugal, driven by a 10-hp. motor. This pump, made by a local foundry, is lined, and has a white-iron open runner. The liners have given an average life of eight months and the runners and shafts of six months; the cost of upkeep of the pump is about \$5 a month. At first, the pumps were driven at 525 rev. per minute and required 6.5 hp. At present, the speed is 460 rev. per minute, taking 5.8 hp., and keeping the pulp all of the same specific gravity with the aid of the 1.2 hp. used for compressed air. The speed can be dropped to 360 rev. per minute with good agitation, but not with the constant specific gravity desired for continuous agitation. There is often a false economy in cutting power costs of grinding or agitation a cent a ton, with a probable consequence of twice that loss in extraction. The actual power used, figured as motor input, is 7 hp. per agitator, or about 1 hp. for each 12 tons of ore or 42 tons of pulp. At the Goldfield Consolidated mill 200 tons of ore in a 1.5 to 1 pulp has been agitated for $7\frac{1}{2}$ hp., or 1 hp. for $26\frac{2}{3}$ tons of ore or $66\frac{2}{3}$ tons of pulp.

Several of the earlier failures of the Trent agitator were due to too small an area of nozzle discharge and to improper sizes and speeds of the pumps used. The following tests will illustrate some of the points governing the Trent agitator. With the agitator equipped with $15\frac{3}{4}$ -in. nozzles, with the pump running at 450 rev. per minute, and with 3 ft. of pulp in the tank, the pressure on the pump discharge was $7\frac{3}{4}$ lb. With the pump stopped, the pressure from the static head on the discharge was $1\frac{1}{2}$ lb. The actual increase of pressure, due to the pump, was $6\frac{1}{4}$ lb. With the pump running and the discharge valve slowly closed, the pressure increased to 15 lb. and the power decreased one-half. With the agitator run under similar conditions, but equipped with $1\frac{1}{2}$ -in. nozzles, giving four times the discharge area, the pressure on the discharge pipe was $2\frac{1}{2}$ lb., or only 1 lb. increase of pressure above the static head; yet the agitator turned at the same speed of 2 rev. per minute and the motor input of 5.6 hp. remained the same. The reason for the power remaining the same is that a centrifugal pump, running at a given speed, takes approximately the same horsepower for a wide range of heads, but the efficiency, it is to be remembered, varies greatly within this same range. The reason for the agitator running at the same speed is that the greater volume pumped in the second case, acting under less pressure, exerted as much turning force as the lesser volume under greater head.

Figuring from nozzle formula, the quantity discharged in the second case was twice that in the first case, giving, therefore, twice the upward

velocity in the tank. The stronger the upward velocity, the more uniform the specific gravity of the pulp. It is evident, therefore, that a large nozzle area, with a pump designed for handling a large volume at a low head and slow speed, is the best combination, and such a pump, made along the lines of the modern Traylor sand pump, should not need repairs for a period of a year. In both the above cases, as the agitator was filled from 3 ft. in depth to 17 ft., the pressure on the pump discharge increased, but nearly in the same amount as the static head, giving the same difference or actual working head, the same power, and the same agitator speed. Consideration of these points takes the successful installation of this agitator and its pump out of the realm of guesswork.

At times, when the mill heads were high, it became necessary to replace pregnant with barren solution. The battery-tank solution fed to the glands of the pumps acted as an increasing diluent in the continuous circuit, but did not remove pregnant solution. The top suction of the agitator is about 4 ft. below the top of the tank, and by cutting off the air to the agitator this top 4 ft. becomes a static thickener, ending abruptly in the moving pulp at the agitator suction line. The pulp in the tank from this line down will have a constant gravity without the aid of the air. A decanting pipe placed in this top 4 ft. of the agitator removed the pregnant solution direct to the silver tank. The decanting of a full 3-in. pipe stream can begin 30 min. after the air is turned off.

With a crowding of tonnage for several months to over 200 tons a day, which was beyond the capacity of the thickeners ahead of the agitators, the overflow of the four 8-ft. Callow cones in the concentrating department was diverted from the thickeners direct to a well, 2 ft. in diameter and 3 ft. deep, in the center of the above agitator, which was No. 2 in the series. This pulp was very thin and contained the lightest particles of the ore; yet, with all this pulp flowing into the top, this agitator delivered the mill flow of ore in a thicker pulp and of a lower value of solution to No. 3 agitator than it received from No. 1, and, while keeping its pulp in constant agitation for 14 ft. of its depth, it delivered a clear stream of pregnant solution to the silver tank. At another time for several months, with less tonnage but a higher grade of ore, No. 4 agitator was fitted with an overflow launder, and a well, 2 ft. in diameter and 4 ft. deep, to take the feed from No. 3 agitator. The agitator was run without air, with top suction closed, bottom suction open, and a bottom discharge to No. 5 agitator. The pulp entered the agitator at $2\frac{1}{2}$ to 1, and was discharged at 1 to 1 to agitator No. 5, where barren solution was added. The extraction in the tank remained the same as before, when it had air, which, in my opinion, was due to the fact that the greatest extraction had taken place by the time it reached this agitator, and the air included in the pulp was sufficient for extraction. I have known a Dorr thickener, placed in the middle of a series of agitators, to make a greater extraction than the

preceding agitator. However, when the air has been cut off No. 1 Trent agitator in our series, only half the usual extraction has been made. The No. 4 agitator pump on this work was cut to 340 rev. per minute, taking 3.5 hp.; yet the agitator started up in its 1.45 pulp after a shutdown as rapidly as the other agitators with only 1.25 pulp. The reason for this is that as the gravity increases the viscosity also increases, hindering the settlement of the heavier sandy particles, which are further handicapped by a diminished difference in gravity between themselves and the surrounding pulp. On this basis, with a proper thickening of the pulp, pyrite concentrate can be readily agitated in a Trent agitator, as is being done at present at the Original Amador Consolidated Mines Co. mill at Amador, Cal.

The next and final step in replacing of solutions was proposed to us by L. C. Trent, the inventor of the agitator. The idea was to feed to the agitator pump a ton of barren solution for each ton of ore fed to the tank, to overflow from the tank the whole tonnage of solution fed to it, and to discharge a 1 to 1 pulp from the bottom of the tank, the solution in which should be the barren solution fed to the pump. This was to be secured by feeding the new pulp to a distributing umbrella on the surface of the pulp in order to feed it in a thin sheet, traveling nearly horizontally, and by having the bottom suction draw pulp from all over the tank by means of a perforated bustle pipe inside the tank, these two doing away with all strong currents. A bypass between the discharge and the suction of the pump would regulate the speed of the agitator, and a series of manifold valves would regulate the flow of barren solution to the pump. A 1 to 1 pulp would be established in the bottom 6 ft. of the tank, and above this a sharp change to thin pulp, and then to clear solution at the top of the tank. The new pulp entering the tank would enter on the surface, only causing a ripple. The solid content would settle through the clear zone into the thick pulp below, but the solution would overflow into the launder, since there would be no downward current, due to the fact that as much solution was introduced at the bottom of the tank as was removed. For a similar reason, the solution fed in at the bottom would find no rising current, and, being mixed with the pulp, would be discharged as an integral part of the discharge pulp. By this method, the inventor reasoned that it would easily be possible to make a 90 per cent. replacement of the dissolved silver entering the tank.

At the time of this visit, every agitator was needed for agitation purposes, the canvas-leaf filter was doing good work, and, lastly, we had little faith in making a high replacement of pregnant solution by this method, for all the many efforts along this line had been failures; and it was generally conceded that the diffusion of solutions of equal gravity took place so quickly and so completely that such a scheme was not practical. However, these replacers were installed at the Gold Cross and

Imperial Reduction Co. plants near Ogilby, Cal., and news of their successful operation came to us not only from the company headquarters, but from the traveling fraternity of millmen looking for a job.

In February, 1915, with a decrease in tonnage, No. 4 agitator was fitted up for a trial as a replacer. With the agitator filled with 1.20 specific gravity pulp, containing 203 tons of solution assaying 7 oz. silver, the feed of pulp and the air were cut off, and the bottom suction opened. Three hours later, when the pulp in the bottom of the tank reached 1.36 specific gravity, a flow of barren solution, assaying 0.13 oz. silver per ton, was admitted to the pump at the rate of about 3 tons an hour. At the end of 12 hr. the gravity at 4 ft. from the bottom was 1.36, at 8 ft., 1.11, and at 12 ft., clear; and the solution assays were 3.94, 4.57, and 6.97 oz., respectively. In 36 hr. there had been added a little over 100 tons of solution, and the specific gravity at 4 ft. from the bottom was 1.34; at 8 ft., cloudy; at 12 ft., cloudy, and the overflow clear; and the solution assays were 1.79, 2.52, 6.21, and 6.6 oz., respectively. Since there was no discharge from the tank, the addition of barren solution had to be slow to keep from thinning the pulp in the bottom of the tank. The replacing action is distinctly shown by the assays. At the end of this time pulp feed and pulp discharge were started into and from the replacer. The results of the next two days' work are shown in Table I.

The sharp reduction in the value of the ore, due to the extraction made in the replacer, makes it difficult to calculate the efficiency of the replacement, but, since most of the ore is in the bottom 6 ft. of the tank in the barren-solution zone, it is probably true that most of the dissolved silver goes out with the discharged solution.

During the second day there was more solution discharged from the tank than barren solution fed into the pump, and the effect is shown by the assays.

On the third day the agitator slowed down, and shutting the bypass did not increase the speed; yet the turning pressure of a man's hands on the ball race would turn the agitator faster. It pointed clearly to a choked suction pipe, which, with an inventor's optimism, was submerged under 12 ft. of pulp. Not being able to continue the experiment, the top suction was opened and the agitator started off at 3 rev. per minute. The result of the experiment demonstrated that a barren zone could be maintained in the bottom of the tank and a high percentage of replacement made, but it also showed that the capacity of the 24-ft. tank was limited to 60 tons of ore per day at a $2\frac{1}{2}$ to 1 dilution, in order to keep a clear overflow, and proved that the mechanical arrangement was defective.

To improve the mechanical defects, the author designed both a new arrangement for the incoming pulp and a new suction for the replacer pump. Instead of the umbrella, a well, 2 ft. in diameter and 4 ft. deep, was placed at the center of the tank around the mast. The bottom was

made solid and the sides perforated with $\frac{3}{4}$ -in. holes. An arm was put on the mast, which kept the bottom of the well clear.

The suction of the pump was carried around the outside of the tank as a bustle pipe, with the diameter proportioned to the flow. Twelve $1\frac{1}{2}$ -in. branches entered the tank at the floor level and ended at varying lengths in upturned ells. Each branch suction was connected to the main pipe by a plug cock, and each had a $\frac{3}{4}$ -in. connection for testing the flow and for connecting high-pressure water if it was choked.

On Mar. 5, 1915, the replacer was started again with these changes. The new well still introduced the incoming pulp horizontally, but below the surface, giving a greater distance down to the slime line at the overflow launder. The new suction made it possible to test and keep open the 12 suction ports at all times, and left nothing inside the tank to give trouble. The agitator, being the new style, was capable of being raised out of the tank for cleaning the nozzles, and for repairs if necessary, and dropped back without affecting the solution zones in the tank. In starting the experiment, the introduction of the barren solution, as before, gradually established a barren zone, or, rather, in this case, a zone of low grade at the bottom, increasing in value upward in the tank. This suggests a method for very small plants, such as used on old dumps, of using the replacer as a thickener first, an agitator second, and lastly as a replacer, discharging after making a second replacement with water. After establishing the barren zone, the feed into and discharge from the replacer were started, and the results obtained are shown in Table I.

A detailed study of the table brings out many interesting features of the replacer. A feed of replacing solution considerably in excess of discharge solution for the period of a shift results in a marked lowering of the solution value in the thick pulp and a decrease in that of the thin pulp, while the opposite condition of feed will increase the value likewise. The sharp change in the silver content of the solution is between the thick 1 to 1 pulp and the thin pulp above it, for the reason that the diffusion in the thin pulp is rapid. Hence, there is no advantage, in installing a replacer, in having any more depth above the 6 ft. of thick pulp than is necessary for the settling of the ore. It is possible to discharge the pulp thicker than 1 to 1, but it takes more power applied to the pump to turn the agitator. With a positive drive overhead, taking very little power, the pulp could be discharged as an 0.8 to 1 pulp or thicker with better replacing results.

In order to calculate the replacement of solutions in these tests, two methods are used as checks: First, dividing the ounces overflowed by the ounces fed; second, dividing the ounces discharged by ounces fed. However, before doing this, the ounces extracted from the ore and the ounces fed in with the replacing solution should be divided between the ounces overflowed and ounces discharged. These two items amount

TABLE I

Time Date Shift			Pulp Feed						Overflow Solution			13 Ft. Valve				
			Spec. Grav.	Ratio Solution to Ore	Ore			Solution			Tons	Oz. per Ton	Total Oz.	Spec. Grav.	Oz. per Ton	
					Tons	Oz. per Ton	Total Oz.	Tons	Oz. per Ton	Total Oz.						
First Test:																
1/28 3-11			1.24	2.25	22.7	4.90	122	51.2	7.91	405	70.0	6.66	466	1.05	5.66	
11-7			1.23	2.37	21.9	4.56	100	51.8	7.41	384	61.0	6.60	402	1.05	5.63	
7-3			1.21	2.63	24.2	4.14	100	63.8	7.02	442	50.0	6.47	323	1.05	5.82	
24 hr.					68.8	4.68	322	166.8	7.42	1,237	181.0	6.58	1,191			
1/29 3-11			1.19	2.95	18.1	4.02	72	53.1	6.82	362	74.0	6.36	471	1.06	6.29	
11-7			1.21	2.63	18.6	4.49	83	49.1	6.88	338	34.0	6.35	216	1.06	6.50	
7-3			1.20	2.78	21.5	4.41	95	59.9	6.82	408	50.0	6.83	341	1.06	6.59	
24 hr.					58.2	4.31	250	162.1	6.83	1,108	158.0	6.50	1,028			
Second Test:																
3/7 24 hr.			1.25	2.15	63.6	2.23	142	108.0	5.62	607	120.0	4.40	530	1.07	4.59	
3/8 7-3			1.25	2.15	22.7			49.0	6.84		50.0			1.09		
3-11			1.22	2.50	21.5			54.0	6.84		51.0			1.09		
11-7			1.23	2.37	18.6			44.0	6.84		45.0			1.11		
24 hr.			1.23	2.34	62.8	2.32	146	147.0	6.84	1,005	146.0	5.20	760		5.24	
3/9 7-3			1.23	2.37	19.0			45.0						1.09		
3-11			1.21	2.63	17.9			42.0						1.10		
11-7			1.23	2.37	16.5			43.0						1.10		
24 hr.					53.4	2.32	124	130.0	6.80	882	155.0	5.60	866		5.72	
3/10 7-3			1.20	2.78	16.8			45.0								
3-11			1.20	2.78	19.1			53.0								
11-7			1.19	2.95	20.6			61.0						1.09		
24 hr.					56.5	2.24	126	159.0	5.66	900	184.0	5.30	970		5.28	
3/11 7-3			1.18	3.13	18.0	2.24	40	56.0	5.62	314	55.0	5.39	296		5.30	
Totals:																
First Test.....					127.0	4.50	572	328.9	7.13	2,345	339.0	6.54	2,219			
Second Test.....					254.3	2.30	585	600.0	6.18	3,708	660.0	5.17	3,416			

to considerable in our tests, because we were not equipped to add barren solution, and, secondly, the ore still lacked sufficient agitation to be ready for filtering. Since the replacing solution is fed to the bottom of the tank, and assays show that it remains there as a partly barren zone, and since over 90 per cent. of the ore is in the bottom half of the tank and a large part of it in contact with the barren replacing solution, it is fair to presume that over 80 per cent. of the replacing solution and ore extraction go out in the discharge and 20 per cent. reaches the overflow.

In the first test, 27 oz. of silver was introduced with the replacing solution, and 112 oz. of silver was yielded to the solutions by the ore, or a total of 139 oz., 20 per cent. of which is 28 oz. and 80 per cent. 111 oz. Silver to the amount of 2,345 oz. was fed with the pulp solution, and of this 2,219 oz. less 28 oz. was overflowed, giving 93.2 per cent. replacement. Figuring by the second method, of the 2,345 oz. of silver fed, only 310 oz. less 111 oz. of it was discharged, giving 91.5 per cent. replacement.

In the second test 43 oz. of silver was introduced with the replacing solutions and 158 oz. of silver derived from the ore. The silver fed with the pulp solution was 3,708 oz. and of this 3,416 oz. less 40 oz. was over-

TABLE I—Continued.

9 Ft. Valve		4 Ft. Valve		Pulp Discharge									Replacing Solution		
				Spec. Grav.	Ratio Solution to Ore	Ore			Solution						
						Spec. Grav.	Os. per Ton	Tons	Os. per Ton	Total Os.	Tons	Os. per Ton	Total Os.	Tons	Os. per Ton
1.29	4.99	1.41	2.41	1.38	1.29	5.1	4.13	21	6.6	2.10	14	14.0	0.20	3.0	
1.23	4.04	1.43	2.34	1.45	1.03	10.2	4.17	42	10.4	2.22	23	18.7	0.20	3.7	
1.07	4.88	1.42	2.40	1.42	1.13	26.5	4.01	106	30.0	2.17	65	20.2	0.23	4.6	
.....						41.8	4.07	169	47.0	2.19	102	52.9	0.21	11.3	
1.06	6.04	1.45	3.00	1.43	1.10	21.4	4.02	87	23.6	2.60	61	16.3	0.27	4.4	
1.07	6.25	1.49	3.20	1.46	1.00	22.9	3.93	90	22.9	2.73	62	15.3	0.29	4.4	
1.09	6.25	1.45	3.10	1.42	1.13	29.5	3.91	114	32.9	2.60	85	25.0	0.29	7.5	
.....						73.8	3.95	291	79.4	2.62	208	56.6		16.3	
1.17	4.44	1.47	1.83	1.46	1.00	43.4	1.70	74	43.4	1.52	66	50.4	0.16	8.0	
1.12		1.45		1.43	1.08	22.8			24.4	1.30	32	25.5			
1.10		1.46		1.46	1.00	24.0			24.0	1.52	36	21.3			
1.11		1.48		1.48	0.94	20.4			19.2	1.58	30	20.2			
.....	4.60		1.91			67.2	1.92	129	67.2	1.46	98	67.0	0.16	10.7	
1.10		1.45		1.46	1.00	19.5			19.5	1.63	32	23.9			
1.12		1.48		1.48	0.94	13.5			12.5	1.72	21	24.0			
1.10		1.48		1.49	0.92	13.3			12.2	1.70	21	22.6			
.....	5.63		2.36			46.3	1.92	89	44.2	1.68	74	70.5	0.15	10.5	
.....				1.48	0.94	13.0			12.2	1.53	19	19.0			
.....				1.47	0.95	20.2			19.0	1.45	27	24.6			
1.10		1.52		1.51	0.87	17.3			15.0	1.67	25	22.0			
.....	5.02		1.80			50.5	1.90	96	46.2		71	65.6	0.16	10.5	
.....	5.00		1.90	1.50	0.89	20.9	1.90	39	18.7	1.37	26	22.0	0.17	3.7	
.....						115.6	3.98	460	126.4	2.46	310	109.5	0.25	27.4	
.....						228.3	1.87	427	219.7	1.52	335	275.5	0.16	43.4	

flowed, giving 93.2 per cent. replacement. Figuring by the second method, of the 3,708 oz. fed, only 335 oz. less 160 oz. of it was discharged, giving 95.3 per cent. replacement.

In actual practice there is always a little silver in the replacing solution, and, as has been found in the continuous-decantation method, there is always an extraction from the ore, no matter how long it has been treated before decantation. These items were not considered in the foregoing calculations, as they were exceptionally high under our conditions. Again, in the tests, 11 per cent. more replacing solution was added than was discharged, which could be done with the final water replacing, due to the building up of solution tonnage.

However, it was established to our satisfaction:

1. That replacing with the Trent apparatus is practicable.
2. That it is easy to control and regulate the replacer; the one important factor being to regulate the solution fed to the pump and that discharged from the replacer. In a plant designed for this system this would be a much easier matter than in our case.
3. That on a quick-settling pulp it would handle a remarkably large

tonnage as far as adding the necessary barren solution is concerned, but that on a pulp carrying a fair percentage of light, flocculent slime it is limited in tonnage to the rate at which the light, flocculent slime will settle in a neutral or no-current tank. In a Dorr thickener there is a downward current to aid settling. At the new mill at Aurora, Nev., the light slime hardly settled in a Dorr thickener, and practically remained in suspension in the Trent replacer.

4. That mechanically the replacer worked very satisfactorily, but that changes could be made to cut down the power consumed and yet make the rotating action more positive.

5. That with a mixture of Tonopah ores, 60 tons of ore in a $2\frac{1}{2}$ to 1 pulp, with 60 tons of replacing solution, was all that a replacer 24 ft. in diameter and 18 ft. high could handle with a clear overflow and a 1 to 1 discharge.

The use of this agitator for replacer experiments necessitated bypassing all but 60 tons past this agitator, and caused a loss in extraction that made it necessary to stop the experiment without a working test extending over a few weeks' time; but the author believes the same results would have continued, as everything about the installation was capable of easy adjustment and repair.

Before changing back to an agitator, the replacer was run as an ordinary thickener with a 1 to 1 discharge, all replacing solution and gland water being cut off the pump, and the speed of the agitator cut to one revolution in 5 min. Hose lines were strung to add a large tonnage of solution to the 60 tons of ore in a $2\frac{1}{2}$ to 1 pulp. To run as a thickener and make 90 per cent. replacement with a 1 to 1 discharge, the 60 tons of ore would have to be diluted to a 10 to 1 pulp by the addition of 450 tons of barren solution to the 150 tons of pregnant solution. About a 7 to 1 pulp was all that could be supplied without interfering with mill operations, but the slime line rose rapidly from 4 ft. below the surface up to the overflow launder. The 60 tons capacity of the replacer had appeared disappointing, but the test showed it to be a good tonnage on this particular ore compared with replacement by dilution; but it also confirmed the author's conviction that on an ore containing considerable flocculent slime the various replacing methods will have a strong rival in the older leaf and pressure filters that make an easily washed cake from the mixture of the flocculent and sandy components of the ore.

The great advantage of the replacer over the thickener-dilution method is in the smaller installation required, and the small amount of barren solution required compared with the large tonnage of barren and partly barren solution of the dilution method.

The main drawback to the replacer system has been, first, lack of faith in its principles, and second, lack of faith in its mechanical features.

To one that has operated the replacer it is easy to understand that in a

1 to 1 and thicker pulp, the pulp is so thick that the ore and solution are approaching a solid, and that the viscosity of the pulp is so great that the solution is not free to move above amid the ore particles, but, rather, is constrained to move with them; and that, once the barren solution is whipped into the pulp by the pump, it tends to remain locked up in it. The pump is of such a size that the difference in specific gravity between its suction and its discharge is only a couple of hundredths, so that the discharge pulp has no tendency to rise in the tank. This explains why the Trent method is a success and why other methods of adding the replacing solution directly into the bottom of a tank have failed.

The last year has brought about a marked improvement in mechanical simplicity and reliability in the mechanism of the replacer. One strong objection to it, compared with the dilution system, is the power required to operate it. However, this does not seem excessive when one considers the saving of the power that is consumed in handling and precipitating the large tonnages of solution required by the dilution methods. The author believes the power consumption of the replacer can be made nearly as low as that required by the equivalent number of thickeners, by the simple device of not depending entirely upon the pump to rotate the agitator, but to drive the agitator from overhead by a worm drive similar to that of the Dorr thickener and to use the pump more as a mixer and circulator of pulp, running it at a slower speed and with larger nozzles on the agitator. Our experience was that it required about 3 hp. on the motor to turn the agitator at only 1 rev. per minute in a 1 to 1 pulp, when a man with a 1-ft. leverage on the mast could turn it with ease. Why, then, should pump nozzle velocity be relied upon to turn the agitator when its proper function should be that of mixing the barren solution into the pulp and giving only the turning effect derived from that function? The pump, with a slower speed and lower working head, should need only annual repairs.

With these modifications, the author believes that the replacer will become an accepted machine in the treatment of slime that is not too colloidal in its nature.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Recovery of Mercury from Amalgamation Tailing, Buffalo Mines, Cobalt

E. B. THORNHILL, * E. M., COBALT, ONT., CANADA,

(San Francisco Meeting, September, 1915)

In this paper on the recovery of mercury as sulphide, from the residues from the amalgamation and cyanide treatment of high-grade ores and concentrates, I will not discuss the many reactions, chemical and other-

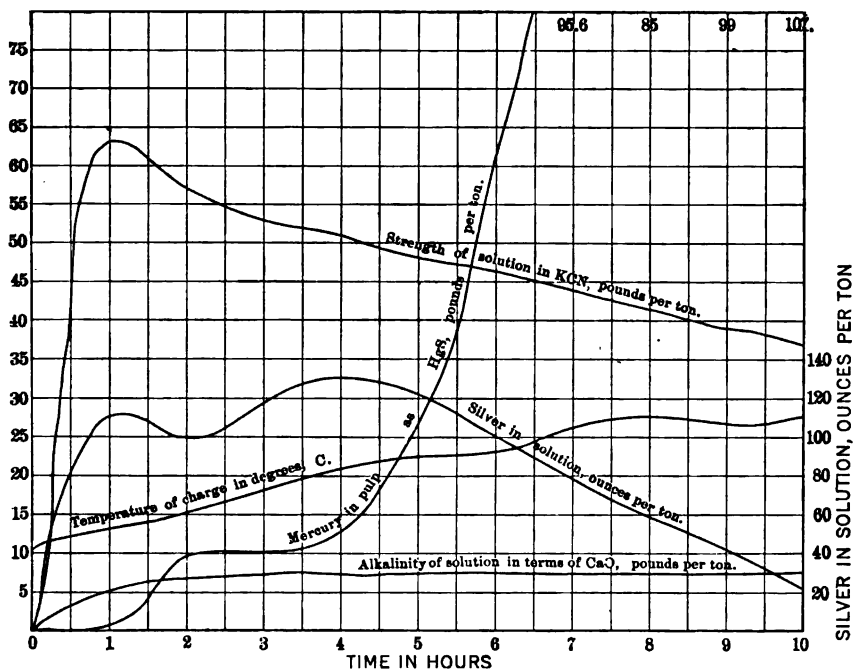


FIG. 1.

wise, that take place in the general process, but confine myself more particularly to the methods of recovering the mercury.

I submit charts of two amalgamation-barrel charges, Figs. 1 and 2, which are self-explanatory. The flow sheet of the high-grade and the mercury extraction plants is shown in Fig. 3.

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In the amalgamation of high-grade silver ores and concentrates in strong cyanide solution, as practiced in the Cobalt district, considerable mercury is retained in the residues from the amalgamation process. The greater part of this mercury is in the form of mercuric sulphide, from 5 to 10 per cent. only of the total mercury content being in the metallic state.

Attempts to eliminate this loss in the amalgamation process were made, but all resulted in a low extraction of the silver in the ore, and attention was then directed to the recovery of the mercury from the residues.

The process developed at the Buffalo Mines, T. R. Jones, General Man-

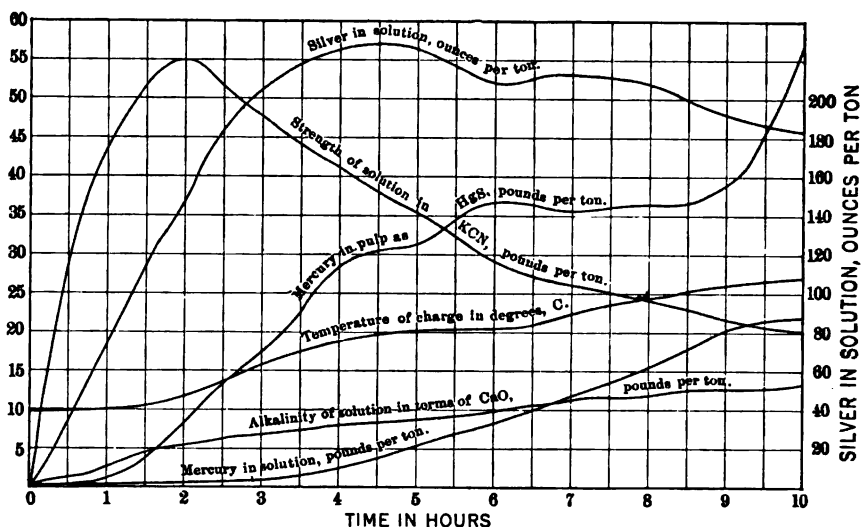


FIG. 2.

ager, for this purpose consists briefly in leaching out the mercuric sulphide with a caustic, alkaline sulphide solution, then precipitating the mercury from solution with metallic aluminum. The equations for solution (1), and precipitation (2) are:



Small-scale experiments showed that a complete extraction of the mercuric sulphide could be made by an 8 to 10 min. treatment of the residues with the alkaline sulphide solution. Advantage was taken of this fact in the commercial plant, by applying the solvent to the residue on the filter leaf, as no agitation of any sort was required.

The operation of the commercial plant is essentially as follows: The residue in the pregnant cyanide solution is caked on a Moore filter

leaf of the usual construction and the cake washed free of silver solution with water. The basket is then lowered into the sodium sulphide

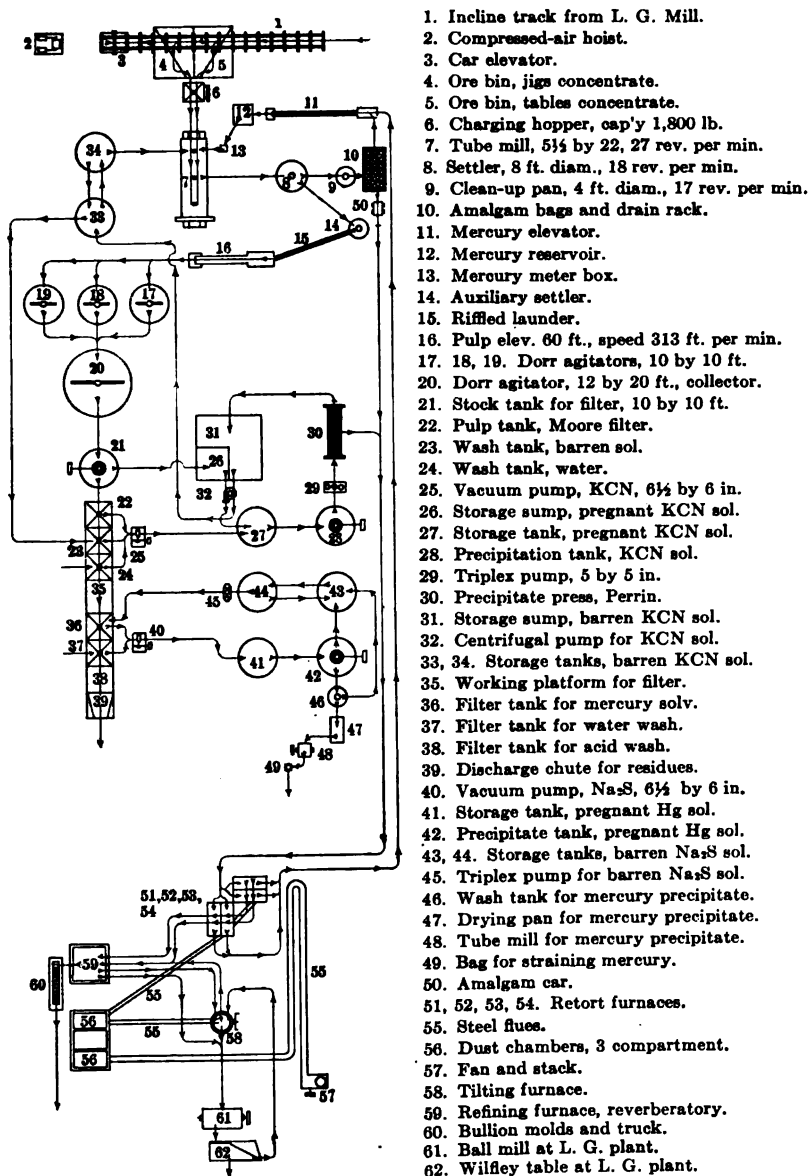


FIG. 3.—FLOW SHEET OF THE HIGH GRADE AND MERCURY EXTRACTION PLANT.

solution and this solution drawn through the cake until the effluent shows only a trace of mercury. Usually 1 ton of solution per ton of residue is sufficient. This mercuric sulphide solution is pumped to a

precipitating tank and the mercury thrown down by adding granular aluminum to the agitated solution. Agitation is then stopped, the precipitate allowed to settle, and the clear solution decanted. The precipitate of mercury is then run into a small wash tank by sweeping it through a hole in the bottom, by means of a raking mechanism, pieces of old rubber belting being riveted to the bottom of the rakes. The precipitate is then washed with water by decantation and drawn off into a steam drying pan. After drying, the fluid mercury and the powdered metallic portion are separated by raking the latter off with a hoe. The fluid is strained through canvas and is ready to return to the circuit. The powdered material, containing approximately 75 per cent. mercury, is then retorted, and the mercury condensed in the usual manner.

The strength of the solvent used is kept up to 4 per cent. sodium sulphide and 1 per cent. sodium hydroxide. Weaker solutions can be used with equally good results, but the quantity of solution required was found to be directly proportional to the strength; that is, with a 4 per cent. sodium sulphide solution, 1 ton of solution would extract the mercuric sulphide from 1 ton of residue, but if a 2 per cent. solution was used, 2 tons would be required per ton of residue treated. The concentrated solution offers the advantage of less solution to handle and an economy of aluminum in the precipitation. The sodium sulphide used is the commercial salt costing \$1.25 per cwt. in barrels F.O.B. cars Toronto.

The precipitant used is a waste product of aluminum casting foundries, containing 75 per cent. aluminum, the impurities consisting of varying proportions of copper, silica, wood, waste, grease, etc., the grease being burned off before using the aluminum. About $\frac{1}{4}$ lb. of this material is used per pound of mercury precipitated.

It was anticipated that the sodium aluminate would accumulate in the solution to such an extent that some special means would have to be taken to remove it. This, however, has not been the case. Some aluminum as the hydrate, falls with the mercury on precipitation, and some is removed during the process of leaching, presumably as calcium aluminate, which collects on the filter cake. This precipitation of the aluminum regenerates caustic soda so that the consumption of this chemical is reduced to $\frac{1}{10}$ lb. per pound of mercury recovered, instead of $\frac{1}{2}$ lb. as shown by the theoretical equation for precipitation. Sodium sulphide is also regenerated in precipitation, but there is also a mechanical loss of approximately 20 per cent. of the solution used in leaching, as no water is used to recover the retained sulphide.

From May, 1914, to March, 1915, 37,650 lb. of mercury have been recovered at a cost of approximately 13c. per pound for labor and chemicals.

The laboratory methods of determining the mercury in the different products of the process may be of interest. To determine the total

mercury (both metallic and sulphide) 0.5 g. of the material is well mixed with cast-iron filings, free from grease, and placed in a hard-glass tube, sealed at one end, with a contraction at about 2 in. from the sealed end. The mercury is distilled off, by heating the bulb containing the charge, and condensed in the tube just beyond the contraction. After the distillation is complete, the contracted portion of the tube is heated, the bulb portion pulled off and the end of the tube sealed. The tube containing the condensed mercury is allowed to cool, filled one-half full of 0.1 per cent. KCN solution, and 10 to 15 100-mg. gold beads added. Each bead will amalgamate with about 1 mg. of mercury. The tube is shaken until all the mercury is amalgamated; the beads are transferred to a small porcelain cup, washed with water, dried with alcohol and weighed. After retorting off the mercury, they are again weighed. From 10 to 12 determinations can be made in one hour by this method and it is accurate enough for control of operations.

When it is necessary to know the proportionate amounts of mercury as metallic and sulphide two 2-g. samples are weighed out. One sample is digested with concentrated nitric acid and the other with a 10 per cent. solution of sodium sulphide. The mercury in the residue from each is then determined by the method just given.

The strength of the sodium sulphide solution is determined by titrating against a standard zinc chloride solution, using sodium nitro prusside as an outside indicator.

The mercury produced by this process is of exceptional purity. A. R. Ledoux & Co. report that 0.25 oz. silver is the only impurity in determinable quantity.

[SUBJECT TO REVISION]

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Amalgamation Tests

BY W. J. SHARWOOD, LEAD, S. D.

(San Francisco Meeting, September, 1915)

THE assay or estimation of the total gold content of an ore presents little difficulty, when circumstances permit of securing a thoroughly representative sample. The general practice has been fairly standardized, and such errors as occur in the results obtained are for the most part attributable to the methods of sampling rather than to those of assaying, while duplicate assays made by skilled workers usually agree closely.

When it is a question of estimating the "free" or amalgamable gold of an ore, as opposed to the total gold, there is but little uniformity in the methods employed by different workers, and often equally little in the results obtained when duplicate determinations are attempted, no satisfactory standard method having been generally adopted. The methods in use are of two general types—"panning" and "amalgamation."

Panning

For the estimation of the value of the visible gold by panning or washing, a weighed sample may be taken, ranging from an ounce to several pounds; a kilogram or 2 lb. is a favorite quantity for laboratory tests, while a prospector often uses a 'full miners' pan, say nearly 20 lb. or 10 kg. In the case of rich quartz, ground to about 30 or 40 mesh, a Mexican horn is convenient, using 1 to 8 oz. of material, and a 6-in. frying pan is favored by many prospectors in dry districts. Bateas, plaques, and dishes of various patterns are used for the same purpose. With these, fairly close results may be obtained by using for ocular comparison standards consisting of weighed amounts of gold of similar character as regards purity, fineness of division, and the general shape of the particles, which standards are preferably obtained from gold of the same locality, or the same deposit, as the ore or gravel being tested. Men working in one particular district sometimes become surprisingly expert in thus judging the value of an ore by the eye, the weights of both ore and gold being guessed. On the other hand, many have been led to overestimate the value of ore and gravel from unfamiliar localities, where the gold has been very finely divided or of unusually high silver content, and a novice may

be easily misled by the presence of such minerals as lead chromate or molybdate. The presence of "sulphurets" is also often a source of difficulty.

When the visible gold obtained by panning is collected and weighed the results are generally low; when the collected pannings are fused and assayed the results may be high as regards "free" gold, owing to the inclusion of a certain amount of concentrate. Sometimes, after concentration by panning, the concentrate is treated with a little mercury or sodium amalgam, which leads to a combination process.

Amalgamation

The amalgamation assay consists in subjecting a weighed sample of ore to the action of mercury, and separating the latter after the gold is presumed to have become amalgamated. Preferably the original ore or "heads," the "tails" after amalgamation, and the mercury which has been separated, are all three assayed, as the results then check one another. It is common, however, to determine only two of these factors, usually the original ore and either the amalgamated gold or the tailing. For some reasons, especially in the case of an ore carrying coarse gold, it is preferable to assay the tailing and determine the amount of gold amalgamated, assuming the original value as the sum of these.

Inasmuch as some free gold is quite invisible, and even coarse gold may be very difficult to detect in the presence of a large proportion of sulphurets or heavy pyritic minerals, the amalgamation method has obvious advantages, especially as the results of panning depend so largely on the operator's judgment and skill in manipulation.

Numerous methods have been published for carrying out the amalgamation assay, but many of the instructions lack precision, leaving so much to the worker's judgment that they are of little use to the inexperienced, and there is a bewildering variety in the quantities of material and times of treatment prescribed. Such directions as "stir for a few hours, or until completely amalgamated," are not infrequent.

Some of the authors recommend agitation, others simple rotation in a bottle; some add water to make a thin pulp, while others specify that it must be thick enough to keep the mercury from settling. Some grind the ore with mercury in an iron mortar till it is reduced to an impalpable powder, others merely rub or stir with a wooden pestle, others again insist on a Wedgwood mortar; one uses as much mercury as ore, or even more, while another recommends less than 1 per cent. A few prescribe grinding with a large proportion of sodium amalgam, endeavoring to secure a maximum amalgamation; most, however, aim to obtain results which shall compare fairly with those realizable in a well-equipped gold mill; some try to imitate pan amalgamation by adding various chemicals

while grinding in an iron mortar. A few recommend preliminary concentration, amalgamating the concentrate, or at least begin by washing off the slime and lighter minerals; others, also using amounts too large for a single operation, divide them, and use the same mercury on several successive charges of a given ore.

Two pieces of special apparatus may be mentioned. Amalgamated copper pans, or copper-bottomed pans with iron rims, are often recommended for use when the tailing and not the amalgam is to be assayed. Their efficiency is increased by silver-plating, and sometimes by adding a rib or riffle on one side to assist in retaining the mercury. They may be made to fairly imitate the conditions of plate amalgamation. Mechanical bateas or pans suspended by springs are sometimes used for the same purpose. The iron arrastre or Buck's mortar is cast with an annular trough in which a muller is revolved by hand; in the best forms the muller is divided into several segments; when used with a fairly large proportion of quicksilver it represents closely the action of an amalgamating pan.

In the main there are two principal schools: One relies on the assay of the tailing and pays comparatively little attention to the quality or quantity of mercury used, but removes it carefully by panning; the other depends on the precise determination of the gold recovered as amalgam, and hence demands the use of mercury which is either free from gold, or contains only a small and known amount.

In determining the gold present in the mercury after its contact with the ore, very great differences also occur in the published directions; these are discussed to some extent under the head of determination of gold and silver in amalgam and mercury.

In the accompanying table an attempt has been made to compare the more important details of procedure recommended by various writers. Some of the typical methods are further summarized. Some of the variations noted undoubtedly arise from differences in the objects aimed at; some of the writers have evidently had in mind a process which could be carried out in the field by any prospector, and most of them appear to contemplate only a few occasional determinations, as the methods usually demand too much personal attention to be of every-day application.

Among all the published descriptions found, apparently only one—that of Guess—refers to the amalgamation assay as a real means of plant control; in this case it was used as a regular working guide to mill practice and as the basis for settlements in treating custom ore. Another paper—that of Warwick—deserves special mention, as, in addition to describing amalgamation tests, it treats with considerable detail the examination of an ore to decide the various forms in which the gold may occur.

Amalgamation Tests for Free Gold in Ore

Authority	Date	Method or Apparatus	Weight of Ore, Grams	Mesh	Weight of Mercury, Grams	Time of Treatment	Extraction Based on Assay of
H. Louis ¹	1893	Ground in mortar.	900 to 1,350	60	30	1 hr.	Mercury (cupelled).
R. W. Leonard ²	1895	Rubbed in mortar.	300 to 3,000	40 to 60	Small amount.	1 hr.	Mercury retorted.
W. H. Merritt ³	1896	Rubbed in miners' pan.	900	60 to 80	30	2 to 4 hr.	Mercury (cupelled).
T. K. Rose ⁴	1896	Ground in mortar.	120	40*	50 to 75	1 1/4 hr.	Mercury (scorified).
A. W. Warwick ⁵	1896	Stirred in mortar.	10 lots of 100	40*	Each 10	1 1/4 hr.	Tailing.
H. S. Gues ⁶	1898	Bottle agitator.	30 to 120	40*	50 to 100	1 hr.	Tailing.
L. Janin, Jr. ⁷	1903	Bottle agitator.	50			1/4 hr. for each passage.	Plate and mercury parted.
E. A. Hersam ⁸	1904	Small copper plate; passed repeatedly.	100		"Globule."		Mercury (scorified or blow-piped).
E. A. Hersam ⁸	1904	Glass U-tube and air lift.	100				Mercury (parted).
W. E. Darrow ⁹	1907	Bottle agitator.	900	60 or 70	6 to 7	1 hr.	Mercury (used in crucible).
A. T. Rose ¹⁰	1910	Rubbed in mortar.	175 to 350		15	1/4 hr.	Preferably tailing.
J. A. Barr ¹¹	1910	Rotated in bottle.	150		200	2 to 3 hr.	Preferably tailing.
J. E. Clennell ¹²	1910	Rolls in bottle.	1,000	100*	20 to 30	1 to 2 hr.	Mercury parted.
W. J. Sharwood ¹³	1911	Bottle agitator.	100		10	1 1/2 to 2 hr.	Mercury (fusion or parting).
A. McA. Johnston ¹⁴	1911	Agitated in mortar (or bottle).	300				Tailing.
A. McA. Johnston ¹⁴	1911	Copper pan.	500 to 1,000	80 to 100	15 to 20	1 hr.	Tailing and mercury (fusion or parting).
E. A. Smith ¹⁵	1913	Ground in mortar.	140 to 200		15		Mercury (distilled).
H. Camer ¹⁶	1731	Concentrated and ground.	180 to 240	80	Spoonful.	Several hours.	Mercury.
E. W. Buskett ¹⁷		Ground in iron mortar.	450 to 1,350	80	Few ounces.	2 to 3 hr.	Mercury and tailing.
E. Ricketts and E. H. Miller ¹⁸		Concentrated and shaken hot in bottle.	Few ounces.	Fine.	30	1 to 3 hr.	Strains mercury, heats and cupels.
H. Van F. Furman ¹⁹		Copper pan.	300 to 500			Some time.	Tailing.
R. W. Lodge ²⁰		Shaken in pan.	300 to 1,000				Tailing and mercury (parted).
R. W. Lodge ²⁰		Shaken vertically in bottle.	2,000 to 5,000				
R. W. Lodge ²⁰		Revolled in bottle.					
R. W. Lodge ²⁰		Ball mill.					

* Or to correspond with milling practice.

¹ Stamp Milling of Gold Ore.

² Trans., xxy, 645 (1895).

³ Trans., xxy, 187 (1896).

⁴ Metallurgy of Gold.

⁵ Journal of the Society of Chemical Industry, vol. xv, No. 3, p. 182 (Mar. 31, 1896).

⁶ Mining and Scientific Press, vol. lxxviii, No. 13, p. 130 (Sept. 28, 1901);

⁷ Journal of the Canadian Mining Institute, vol. i, p. 10 (1898).

⁸ Pacific Coast Miner, vol. vii, No. 2, p. 30 (Jan. 10, 1903).

⁹ Trans., xxy, 399 (1904).

¹⁰ Mining and Scientific Press, vol. xov, No. 10, p. 300 (Sept. 7, 1907).

¹¹ Mining World (Chicago), vol. xxxii, No. 6, p. 319 (Feb. 5, 1910).

¹² Treatise for Metallurgical Processes.

¹³ Cyanide Handbook (1910).

¹⁴ Patton's Manual of Fire Assaying, 2d ed. (1911).

¹⁵ Rand Metallurgical Practice, vol. i (1912).

¹⁶ Sampling and Assay of the Precious Metals (1913).

¹⁷ Elements of the Art of Assaying Metals (Mortimer's translation, 1764).

¹⁸ Fire Assaying (1907).

¹⁹ Notes on Assaying.

²⁰ Manual of Practical Assaying (1893).

²¹ Notes on Assaying (1904).

E. A. HERSAM suggests two methods:

1. A very small piece of amalgamated copper foil is repeatedly subjected to the action of a stream of ore particles passed over it under the surface of water in a flask, the whole sample being passed over it a definite number of times, each passage taking half an hour. The copper plate is then treated with nitric acid and the gold recovered.

2. A glass apparatus with an air-lift device causes the ore, mixed with water to a thin pulp, to circulate over a globule of mercury in a narrow U-tube. The mercury is finally volatilized; the residue is fused before the blowpipe, or may be scorified, cupelled, and parted.

R. W. LEONARD divides a large sample into several lots which are rubbed successively in a mortar, the mercury separated from the first being used again for the second, and so on. Finally it is cleaned and put into a small iron retort and heated to expel the mercury. The gold is recovered by fusing a little test lead in the retort; the lead is then cupelled, "preferably with the blowpipe."

W. H. MERRITT uses about 2 lb. (900 g.) of ore crushed to 40 or 60 mesh, and rubs this with an ounce (30 g.) of mercury in a miners' pan with a wooden pestle. The mercury is "retorted" in a cup-shaped piece of sheet iron, or in a potato, and the gold weighed.

A. McARTHUR JOHNSTON.—Weigh out 10 assay tons into a Wedgwood mortar and add water to make a thin pulp. Into this place 10 g. pure redistilled mercury and agitate half an hour. No attrition should be allowed other than that between the particles themselves, and between them and the sides of the mortar. Should the ore contain any ingredient tending to foul the mercury some reagent may be added (sodium hydroxide or hydrochloric acid) to make the mercury more active. If rusty gold is present it is often expedient to add sulphuric acid. The pulp could also be agitated with mercury in a bottle, but this method causes flouring or fine subdivision of the mercury and makes its collection very difficult.

To recover the mercury, pan off the sand and concentrate, taking care not to lose minute globules; even with care a slight loss of mercury ensues. A separating funnel may also be used. The collected mercury is cleaned and weighed, and assayed for gold. The gold thus found is multiplied by the original weight, and divided by the recovered weight, to give the total gold recoverable by amalgamation; the result is calculated per ton of ore.

W. H. COGHILL also recommends the use of a small separatory funnel with two stopcocks, such as is used in mineralogical work with heavy liquids, for removing mercury from amalgamated tailing.

A. W. WARWICK places in a porcelain mortar 4 assay tons (120 g.) of ore crushed to a suitable size (or treats several such samples at different meshes), adds water enough to make a very thin paste, then 50 to 75 g. of mercury, accurately weighed. The mixture is stirred, not rubbed, with a light pestle of hard wood for $1\frac{1}{2}$ hr. It is then diluted, the thin paste poured off, and the mercury washed, dried, and weighed. The loss is indicative of the tendency to "flour" and Warwick considers amalgamation impracticable if it exceeds 1 per cent. The collected mercury is placed in a scorifier, covered with a second scorifier, placed in a muffle to vaporize, and the residue scorified with lead, cupelled, etc.

H. S. GUESS describes a method used practically for the control of a custom mill, the ore being crushed to 40 mesh, or to agree with the mill practice. From three to ten samples of 100 g. each are put in wide-mouthed stoppered bottles, with 10 g. of mercury and water, and agitated 30 min. A vertical agitator is used, having a 2-in. eccentric and run at 350 rev. per minute. He pans and discards the mercury, using a deep aluminum pan 10 in. in diameter. The concentrate is separated, dried, and assayed; the tailing and slime are dried together and assayed. The values of concentrate and total tailing are subtracted from the original assay to give the amount amal-

gamated. The returns of a number of mill cleanups are quoted to show their close agreement with these tests.

L. JANIN, JR. uses a much larger proportion of mercury than the preceding. He shakes 1 to 4 assay tons of ore with 50 to 100 g. of mercury in a bottle for an hour, by means of a "milk shake" or Gay-Lussac silver-assay apparatus. The bottle is then inverted, most of the mercury poured out, and the remaining globules separated by the use of a copper pan or dish. The proportion of free gold is determined by an assay of the original pulp and final tailing.

HERMAN CRAMER described in *Elements of the Art of Assaying* the process of amalgamating previously concentrated ores of native silver (or gold) in an iron mortar, using a wooden pestle and a large proportion of mercury, also adding vinegar and alum.¹¹ The mercury is finally to be squeezed through thin leather and the solid amalgam distilled in a glass retort on a sand bath. He mentions the danger of precious metal passing into the receiver with the mercury when the fire is too strong. He adds the following caution (following Mortimer's translation): "If, for want of an apparatus for scorification and coppelling, you have a Mind to indicate by this method, the Quantity of Silver contained in the Body washed; in this case the whole Amalgama must be distilled through the Retort; because Part of the Silver and gold gets through the Leather: Nay, there remains nothing at all of the Silver or Gold within the Leather, if you use too great a Quantity of Mercury, to extract a small Quantity of these Metals: unless the Mercury be saturated with them, by a like previous Process; and even then, you may be easily deceived as to the Quantity and Quality of the Metal." . . . "Amalgamation itself is more proper to Gold than to Silver . . . A true metallic State is required for an Amalgamation, because no Extraction can otherwise be made by Mercury."

The quotations from Cramer have been introduced to show that he was alive to sources of error that have been overlooked by some much more recent chemists.

The Estimation of Gold and Silver in Mercury and Amalgam, and the Preparation of Pure Mercury

These subjects are so intimately connected that they are best treated together. It is a common delusion that only a negligible amount of gold or silver passes with the mercury through leather or cloth used in straining amalgam, and a still commoner one that mercury is completely freed from gold by distillation or "retorting."

At the ordinary room temperature mercury dissolves at least 0.03 per cent. and up to 0.05 per cent. of gold, and I have found about 0.02 per cent. gold and half as much silver in squeezed mercury which had deposited most of its contained amalgam as crystals after long standing at a temperature just below the freezing point of water. The usually quoted value of 0.03 per cent. or 300 parts per million is equivalent to \$180 per ton. The actual content is often much higher, owing either to minute particles of suspended amalgam passing through holes in the straining cloth, or to straining at a higher temperature and retention of the dissolved amalgam, apparently as a supersaturated solution.

If the attempt is made, therefore, to separate the collected gold from

¹¹ This combination of vinegar and alum is also mentioned by Ercker.

the mercury in the amalgamation assay by squeezing through buckskin, it is easy to see that the results are entirely misleading. If we use only 5 g. of mercury with 100 g. of ore (or $\frac{3}{4}$ oz. per pound) at least \$9 may be extracted per ton of ore and pass through the leather with the mercury. This method is entirely inadmissible except for the roughest estimation of coarse gold.

Even when the method of assaying the tailing is adopted, the use of squeezed mercury may lead to serious error; the retention of $\frac{1}{2}$ g. in the floured condition in 100 g. of ore making the tailing too high by at least 90c. per ton.

In retorting mill amalgam an appreciable, though small, amount of gold is carried over with the quicksilver, probably owing to spurting or violent boiling. H. Louis estimates that under the most favorable working conditions at least 5 parts of gold pass over per million of mercury condensed, so that the mercury carries at least \$3 worth of gold per ton. Often much more may be found in it. If mercury carrying \$3 per ton is used in tests, at the rate of 20 g. per 100 g. of ore, the assay of the mercury will indicate a recovery which is too high by 60c. per ton of ore. If the extraction is calculated on the assay of the tailing the error thus introduced is inappreciable unless flouring is excessive.

By a cautious slow redistillation of this nearly pure quicksilver in a clean new iron retort, with a condenser of new iron pipe, it is obtainable practically free from gold. This is the best method to adopt where practicable. Vacuum apparatus may also be used for effecting the distillation under reduced pressure at a much lower temperature than is otherwise practicable, or glass condensers may be used, but neither is of much advantage for the present purpose. Hulett has shown that, if distillation is slow enough to avoid actual boiling, even such volatile metals as zinc and cadmium are carried over in only minute amounts.

It is still easier, at mills where quicksilver is bought in large quantities and comes directly from the mines, to test several of the best looking flasks for gold, and select the best found. Sometimes scarcely a trace is present in new quicksilver; occasionally a flask is found containing a considerable proportion of gold, possibly owing to the use of old mill flasks.

For the preparation of small amounts of gold-free mercury Darrow recommends treating a quantity with an insufficiency of nitric acid, so as to leave the gold in the undissolved portion. The filtered solution of nitrate is then precipitated by means of pure copper, such as electric wire. This is only suited to small amounts. Electrolysis of the nitrate is not practicable, as even the densest graphite electrodes are rapidly disintegrated. A simple method of removing gold from mercury was described by W. Bettel,²² who puts a thin layer in a shallow dish and covers it with a 2 per cent. solution of potassium cyanide, adding a little sodium per-

²² *Mining and Scientific Press*, vol. xcvii, No. 26, p. 881, (Dec. 26, 1908).

oxide at intervals. The water requires renewal occasionally. I have found this effective but rather slow; stoneware or agateware pans may be used, but are somewhat attacked by the alkali.

In estimating the precious metal in small amounts of amalgam or in mercury, two methods naturally suggest themselves. One is to distill off the mercury, the other to remove it with nitric acid.

In practice it is found that the first invariably causes a certain loss of precious metal, while in acid parting the silver is always too low. It is recommended that the acid method be adhered to, so far as gold is concerned.

In case it is desirable to estimate silver also, it is recommended to take two portions of the material. In one the mercury is removed by nitric acid, finishing with hot and fairly concentrated acid; the remaining gold being then cautiously dried and heated to expel any residual mercury, inquarted, and parted as usual. The second portion is put into a small scorifier, covered with another inverted scorifier, and set in a comparatively cool muffle, which is then slowly heated to strong redness, taking care that there is a strong draft to carry the fumes outside.²³ The residue in the scorifier is then covered with a layer of test lead, which is scorified a few minutes and cupelled. The scorifier used as a cover must be carefully examined for traces of adhering metal, and in exact work it is preferable to also scorify a little lead in the cover and add it to the main lot. Small lots of clean amalgam may be similarly treated in a covered cupel to avoid scorifying.²⁴ From the total gold obtained by the wet method, and the ratio of gold to silver indicated by the fire assay, the amount of silver may be accurately determined; such refinements as the correction for cupel absorption being introduced if warranted.

A satisfactory method of recovering both silver and gold from amalgam or mercury is to heat it in a crucible under a heavy cover of assay flux, which is then fused and the resulting lead button cupelled. Thus A. McArthur Johnston recommends a layer of 60 g. of litharge, mixed with enough reducer to yield 25 or 30 g. of lead, and a cover of borax. Ordinary assay flux, with a little silica added, appears to work well. In any case extreme care must be taken to guard against the escape of mercury fumes into the laboratory.

²³ The vapor of mercury has so high a density that a moderate draft is insufficient to carry it off, and heating in a muffle without proper precaution may lead to serious consequences. In regard to the danger of vaporizing mercury without using a condenser, the warning given by Pettus in his translation of Ercker's work may be appropriately quoted: "Take heed lest the *smoak* or vapour go not into thy *Belly*, because it is a *poysonous* and cold *Vapour*, which lameth and killeth: for, he will find that it will there congeal and afterwards spoil his body."

²⁴ It is scarcely necessary to add that the cupellation of amalgam directly, or of gold containing more than a trace of mercury, causes the "spitting" of the cupelling lead, and the contamination of other assays.

For rapidly obtaining the gold from a small amount of amalgam the most convenient plan is to heat it in a miniature retort, made by closing a piece of glass tubing at one end and bending it to a slightly acute angle. The retort is very gradually heated to full redness and then quenched with water.

The device of placing it in a hollowed potato, or on an iron plate covered with half a potato, and heating till the potato is charred, is fairly satisfactory if the amount of gold is not too small.

When solid amalgam is treated with nitric acid a considerable amount of mercury is retained, even after long heating with strong acid, the gold protecting the mercury as it does silver. In the case of mercury containing small proportions of dissolved gold heating with moderately strong acid leaves the gold nearly pure, so that the dried gold can be safely heated without spurting and the last trace of mercury thus removed.

Extremely sloppy or dilute amalgam is very difficult to sample, as it is impossible to secure a uniform mixture of the solid and liquid portions. In sampling such material it is advisable to strain it through cloth at a rather low temperature, and weigh both solid and liquid portions. These can be assayed separately and the original content calculated, or similar aliquots can be taken from each portion and mixed for assay.

A Suggested Standard Amalgamation Test

A satisfactory laboratory method for the amalgamation test must secure the following conditions: The individual attention necessary in the separation of mercury must be reduced to a minimum, and the entire manipulation must be as simple as possible. To this end a mechanical agitator must be used, and preferably one which allows of the bottles being introduced or removed while in motion; to avoid mechanical loss bottles with some rapid but tight closing device must be adopted. Time is saved on the whole by assaying the mercury rather than the tailing, provided sufficiently pure mercury can be prepared or obtained. Sufficient mercury must be used to allow of quick action and to avoid serious error in case a few centigrams are lost. The addition of chemicals is generally to be avoided, and sodium amalgam should be used only when absolutely necessary. The cost of materials and labor must not be excessive.

The method finally to be described was evolved after much experimentation with the above considerations in view. It was intended purely as a laboratory method, the shaker requiring power; it may, however, be used at any assay office, and at a pinch can be carried out wherever nitric acid and a blowpipe are available, as two bottles can be shaken at a time by hand. The cost for materials per assay, including gasoline or fuel oil, is well under 10c.

A convenient agitator is a stoutly built wooden box with 24 or more

compartments, each 3 in. square and 6 in. deep, driven from a shaft running about 250 rev. per minute, giving a horizontal travel of about 2 in. The entire bottom, or each section, should be covered with heavy belting or linoleum. The throw may be varied by means of a wrist-pin adjustable in several holes in a disk. The box may be supported by four vertical springs of steel, $\frac{1}{8}$ by $1\frac{1}{2}$ in., 20 to 24 in. long, bolted to it and to two sills. One of this description has been in use over five years without repairs except for the replacing of a few bolts. The connecting rod is of oak, 2 by $\frac{3}{4}$ in. Where a Frue vanner is available, or any similar mechanism making 200 to 270 horizontal oscillations per minute of about 2 in., it may be utilized for this purpose. In the case of a Frue vanner this is best done by attaching supports to the oscillating frame and carrying the bottle rack across the belt.

The average percentage amalgamated in some thousands of 100-g. tests differs very little from the average stamp-mill percentage of recovery for the same period of years. Provided the samples are ground fairly fine—at least to pass 80 mesh—the results, so far as my experience goes, have differed but little with the fineness, even if ground to 200 mesh. Presumably, with the ores thus examined, the free gold is mostly exposed by a moderate degree of crushing, occurring largely, as is the case with so many gold ores, along former fracture surfaces.

This agitator and the magnesium citrate bottles have been found exceedingly useful for such work as preliminary cyanide tests; when the capacity of the bottle is too small it is convenient to take a number of portions and either make parallel tests or mix the products for assay.

Equipment Required for Systematic Amalgamation Tests

Agitator with 24 compartments, as described above.

Twenty-four "citrate magnesia" bottles, with spring clamps and rubber washers. These have a capacity about 350 to 370 cc.

Twenty-four Griffin beakers, capacity about 250 cc.

Two enameled-iron pans, rectangular, with straight sides, to hold 12 beakers each.

Filtering rack with 12 $2\frac{3}{4}$ -in. funnels and 12 extra beakers.

Hot plate.

Glass-stoppered burette for mercury, standing over an enameled pan.

Robervahl scale, turning with $\frac{1}{2}$ g.

Weighing scoop and steep-sided copper funnel or hopper for charging ore into bottles.

Cylinder, graduated to 100, 150, and 200 cc.

Six-inch funnel for water.

A few porcelain dishes about 3 in. diameter, and some enameled pans about 8 in. diameter and about 2 in. deep.

Filter paper, 12.5 cm., S. and S. No. 595 or similar quality.

Scorifiers, 2 in. or less in diameter; these may be used repeatedly and are best glazed before first using.

Washbottle.

Test lead.

Nitric acid, diluted with about double its volume of water.

Mercury, pure or previously assayed.

Silver foil, cut in pieces of about $\frac{1}{2}$ g. each.

Routine Method for Gold Only

One hundred grams of crushed ore or tailing are weighed out, charged into a bottle by means of the metal funnel, 150 cc. of water added, then 1.5 to 2 cc. of pure mercury from a burette. The stopper is then clamped, the bottle folded in a thick piece of cloth or flannel and put into the moving "shaker" for 2 hr. It is then removed, opened, covered with the thumb, shaken and inverted over a 3-in. porcelain dish, and as much as possible of the clean mercury allowed to run out. It is then shaken again for a moment, and if necessary a little more water is added, and more mercury allowed to run out into a second dish, and so on, as long as any mercury comes out. If not floured the mercury is nearly all removed in two operations. The clean mercury thus obtained is transferred to a beaker. The bottle is again shaken well and inverted so as to let a little sand run out into the dish; this is then panned into an enamel dish. Usually only a few globules are obtained; if much is found the whole charge must be panned. If "floured" a small globule of liquid sodium amalgam should be added at this point.

To the mercury, after collecting in a 250-cc. beaker, is added $\frac{1}{2}$ g. of pure silver foil. If this has been carefully prepared and cleaned by treating with cyanide solution or weak nitric acid, or by slightly amalgamating the surface, it may be used to pick up the small globules of mercury obtained in panning. Nitric acid (about 150 cc. of sp. gr. 1.14 or 1.15) previously warmed to about 70° C. is now poured into the beaker; all the beakers are set together in a pan, which is placed on the hot plate and left until all the mercury has dissolved. If not too hot it is unnecessary to use covers. As soon as the mercury disappears the liquid is filtered, the residue is rinsed on to the paper, washed once or twice with very dilute nitric acid (not over 5 per cent.) and once with water. The paper is then sprinkled with test lead, folded, and placed in a small glazed scorifier, and covered with lead enough to make up the total to 15 or 20 g. Enough pure silver is added to insure ready parting, and 2 or 3 g. of borax glass. The scorifiers are then charged into a muffle, the paper carefully burned, the lead scorified a few minutes, poured, and cupelled. The cupel buttons are parted in porcelain cups, and the gold annealed and weighed.

A correction is subtracted for gold contained in the silver and mercury.

Using a 100-g. sample, each milligram of gold represents 0.29166 oz., or \$6.03 amalgamable gold per ton.

Notes

If preferred 4 assay tons may be used, when each milligram indicates exactly 0.25 troy ounce; or 120.5 g., when each milligram indicates exactly \$5 per ton. With these amounts the water must be proportionally increased, and the manipulation in the bottles is not quite as easy as with 100 g.

In ordinary routine work the tailings are discarded. For special purposes, as when they are to be tested by cyaniding, concentration, etc., or if the original assays are unsatisfactory by reason of the presence of coarse gold, they are carefully panned free of mercury and caught in one of the larger pans, allowed to settle completely, decanted, and if necessary dried for further tests or for assay.

In special work, and particularly in case there is a tendency to flouring, the mercury is weighed out to the nearest centigram (any sodium amalgam being similarly weighed and added to it), and the recovered mercury is also weighed after drying. The recovered gold, multiplied by the former weight, and divided by the latter, gives the total gold amalgamated. In ordinary work with fine-ground ore the loss of mercury runs between 1 and 2 per cent.; and is less if the entire amount of tailing is panned.

The ore should be ground to a fairly constant degree of fineness. The product of a disk grinder, set for about 100 mesh for use in assay work, has been found to give sufficiently uniform results for most purposes. For careful comparative tests it would be preferable to sift to a definite size.

Using 100 g., 150 cc. water is preferable to 200 cc.; with less water the pulp is too thick. Material of unusually high or low specific gravity may require special attention as regards thickness of pulp. Reducing the time from 2 hr. to 1½ hr. *generally* gives good results; a less time is not safe; 1¾ to 2 hr. appear to be safe limits.

The temperature should be nearly constant from day to day; if much above the normal the results are somewhat higher; if it falls nearly to the freezing point they may be several per cent. low.

The concentration of acid recommended (one volume of concentrated nitric acid to two of water, giving about sp. gr. 1.14) was found advantageous after a number of tests. It allows of warming in advance so as to expedite action, without becoming too violent at any stage. The amount recommended leaves a sufficient excess to prevent separation of mercury salts. If concentrated during the heating it may be necessary to dilute slightly before filtering.

Addition of silver to the mercury reduces the time required for solution by about one-half, as determined by a large number of comparative tests. Commercial proof silver invariably contains gold; samples examined have carried about 0.01 mg. per half gram; or about the same amount as found in 20 g. of some of the best samples of mercury tested.

The correction for gold in mercury and silver is best determined by taking five times the amount to be used per assay, parting them together and carrying out the whole procedure exactly as in the regular work, but using five times the standard amount of test lead, scorifying off the greater portion. To avoid repeating this blank test unnecessarily it is well to have a considerable reserve stock of both mercury and silver. A thousand determinations using 1.5 to 2 cc. (20 to 27 g.) require 45 to 60 lb. of mercury, and about 16 oz. of silver.

With ores carrying less than \$3 gold per ton it is advisable to weigh out two or more parallel lots and combine the gold after parting.

If amalgamable silver is to be determined in the ore it is best to make duplicate tests, parting one and scorifying the other, as indicated on a previous page.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

A Rule Governing Cupellation Losses

BY W. J. SHARWOOD, LEAD, S. D.

(San Francisco Meeting, September, 1915)

CUPELLATION is well known to be one of the most effective methods of separating silver and gold from base metals and other impurities, as well as one of the most accurate means for their estimation. In the latter application it consists in absorbing in some porous medium the fused lead oxide formed by the oxidation of the lead "bath," which carries with it the oxides of other base metals present.

It has long been recognized that small amounts of both silver and gold were removed and absorbed together with the base metals, as well as being to some slight extent volatilized, and in work of the highest accuracy a correction is made for the "cupellation loss."

Thus in the assay of bullion this is done by means of "proofs" of like composition, which are manipulated throughout in precisely the same manner as the assay pieces, and are assumed to lose or gain like amounts. In the case of certain very rich ores and commercial products the loss is adjusted by assaying the cupel itself and recovering the precious metal absorbed, but this doubles the labor and does not remedy the loss by volatilization. Arbitrary tables have been drawn up by various authorities with the object of enabling a correction to be applied without the use of a check. These have been of very limited practical use owing to the fact that a number of factors affect the losses sustained.

For a fixed amount of precious metal the loss varies with the amount of lead used, with the nature and amount of the impurities, with the porosity of the cupel, with the air supply, and above all with the temperature at which cupellation is carried on. Pure silver loses relatively much more than gold, but the loss in either is diminished by an addition of the other metal. The concentration of precious metal in the litharge increases as the concentration in the lead increases, but little work has been published connecting the two. If all other conditions remain the same the actual total loss increases, but the percentage loss diminishes, with an increase in the weight of precious metal treated.

The results of an immense number of experiments have been published at various times during the past hundred years giving the actual or the percentage losses suffered under various conditions of cupellation, but the data are widely scattered and few attempts have been made to correlate them, so that no definite law has heretofore been shown to exist

governing the relation of weight and loss, although Fulton has published curves covering certain conditions. If such a law were known it would enable a correction to be applied to a button of any weight, by a calculation applied to the loss observed in a proof of an entirely different weight, but cupelled at the same time under the same conditions. It is believed that such a rule is now available.

This empiric rule may be thus enunciated: *When a Given Amount of Silver (or of Gold) is Cupelled with a Given Amount of Lead, under a Fixed Set of Conditions as to Temperature, etc., the Apparent Loss of Weight Sustained by the Precious Metal is Directly Proportional to the Surface of the Button of Fine Metal Remaining.*

Probably the calculation should be based on the original weight of precious metal taken, but in practice we have to depend on the weight remaining. Variations in the amount of lead have comparatively little influence, and temperature conditions in a given row across a muffle are nearly uniform, so that for practical purposes the proportionality holds good for any one row in a cupellation run, provided the amounts of lead do not differ extremely. Most other variations have smaller effects. If the above is true the following must also be true.

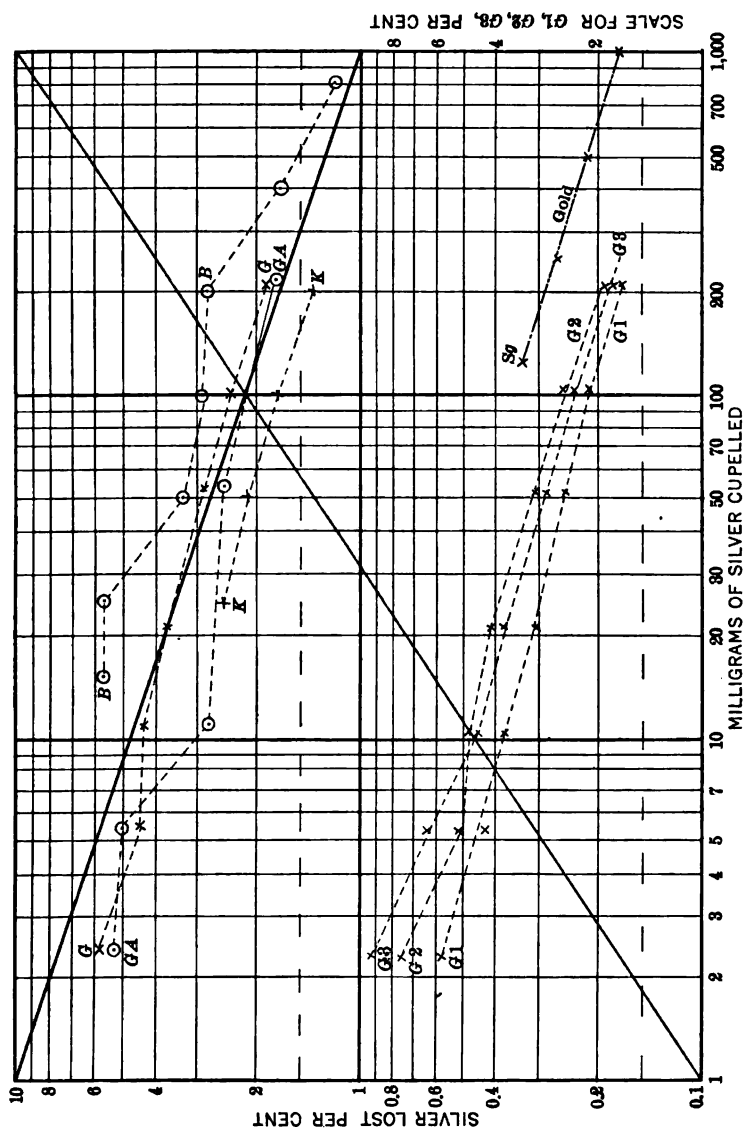
The Loss of Weight Varies as the $\frac{3}{2}$ Power of the Weight, or as the Square of the Diameter of the Button.

The Percentage Loss Varies Inversely as the Diameter of the Button, or Inversely as the Cube Root of the Weight.

Examination of a large number of experimental results, as well as the published data of others, proves that the last proposition is a close approximation to the truth, and therefore proves the others. The readiest proof of the rule, which avoids all calculation, is by the aid of logarithmic paper—that is, paper on which the co-ordinates are plotted on a scale like that of a slide rule.¹ In place of using the ruled paper the distances may be actually laid off by means of a slide-rule scale. When plotted in this way the graph of any equation of the character $y = Cx^n$ appears as a straight line, and all such lines for a given value of the power n are parallel. If the horizontal and vertical scales are equal (as in Figs. 1 and 2) the tangent made by the line with the x axis is the exponent n .

If therefore we plot on the x axis the weights of a series of silver buttons cupelled under precisely similar conditions, and the corresponding losses on the y axis, and the corresponding points fall approximately on a straight line, then we know that y varies as some power of x . Drawing a line parallel to the general run of these points from the point whose co-ordinates are $y = 1$, $x = 1$ to $x = 10$ or 100 , will indicate at once what the value of n actually is.

¹ Multiple logarithmic paper may be obtained at a very moderate price through the Department of Civil Engineering, University of Wisconsin.



G = Godshall, average of three series.

GA = Godshall, average of Tables A and B.

G1, G2, G3 = First, second, and third rows in muffle of Series G.

The percentage losses alone are given, this being the form used in most publications. Godshall's results for various muffle rows are placed for clearness in the lower portion of the diagram.

Fig. 1.

K = Kaufman.

B = Beringer.

Sg = Sharwood (gold).

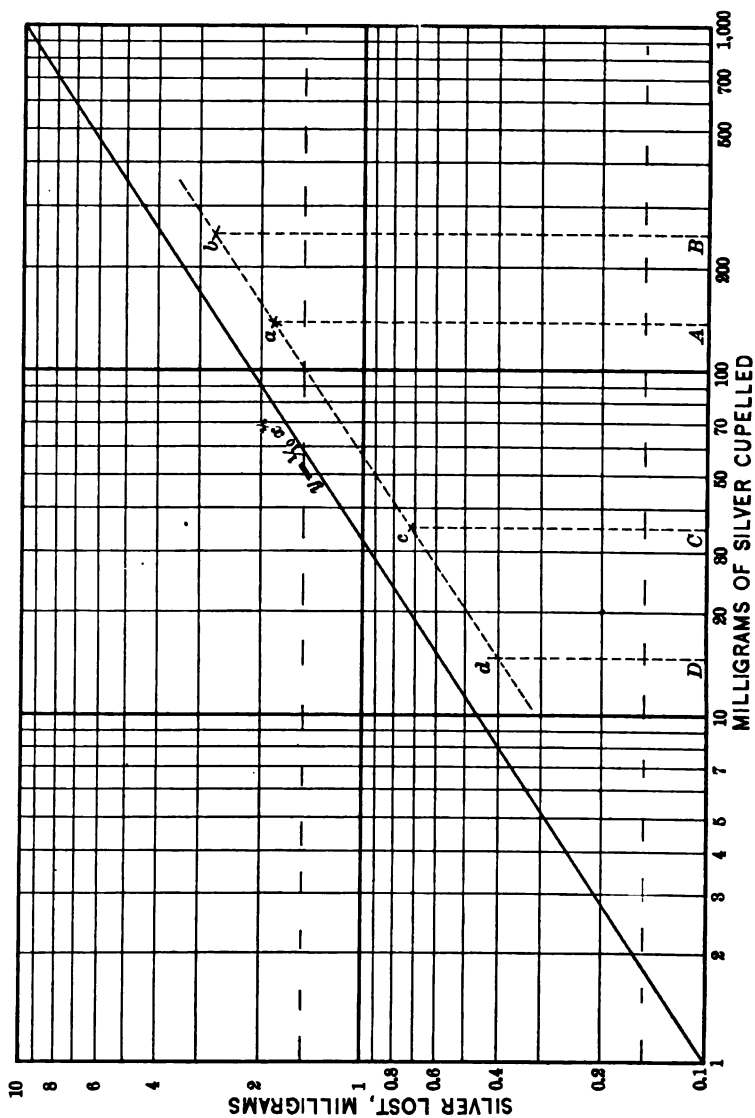


Fig. 2.

If any one will take the trouble to average and plot a series of cupel losses on buttons of various sizes carefully cupelled in the same row, he will, I think, find that they will fall nearly on² lines parallel to

$$y_1 \text{ (weight lost)} = x^{2/3}$$

and

$$y_2 \text{ (percentage lost)} = x^{-1/3}$$

The distances between the parallel lines obtained in various series simply correspond to the values of the constant C for different conditions of temperature, muffle draft, etc.

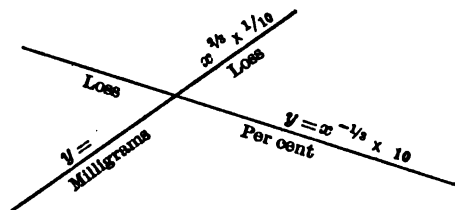


FIG. 3.

A few results are shown thus plotted in Fig. 1.

I have found that such series as those published by Klasek, Godshall, Beringer, and Kauffman (omitting certain erratic ones), and a transcription of the curves plotted by Fulton, all give points falling nearly into general parallelism with the lines indicated, and my own values for silver alone and gold alone agree well except for the smaller values, where irregularity naturally occurs. This rule is the dominant factor in cupellation loss, temperature alone excepted.

As to the possible application of this rule: An approximate correction may be obtained for an ore or the like, if one proof, as near the expected weight as possible, is cupelled in each row, and its exact loss noted. The corresponding point is plotted on the diagram (a in Fig. 2, A representing the final weight), and a straightedge is placed on the diagram, or a line drawn through a , parallel to the guide line, $y = x^{2/3}$. Noting the points on the line or straightedge corresponding to the weights B, C, D , of other buttons weighed, the correction for each is read at once from the scale. In the example plotted in Fig. 2 a button of 140 mg. weight has lost 1.8 mg.; from this it appears that other buttons of 250, 35, and 15 mg. will respectively have lost about 2.85, 0.72, and 0.40 mg.

In a future paper I hope to present some further data on the cupellation of mixed metals, their relative rates of removal, and their concentration in various portions of the cupel.

² In the case of the smaller weights, say those below 10 mg., a considerable deviation from a straight line will probably be observed, unless a large number are averaged, as the relative error due to a difference of a few hundredths of a milligram then becomes serious.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Churn-Drilling Costs, Sacramento Hill

BY ARTHUR NOTMAN,* M. E., BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

SACRAMENTO HILL is a mass of granite porphyry intruded along a fault between Paleozoic sediments and pre-Cambrian schists in the Bisbee district, Cochise County, Arizona.

The intrusion invaded both walls of the fault to a large extent, and the accompanying extensive mineralization of the Paleozoic limestones is the locus of the great high-grade copper orebodies of the Bisbee mines. Prior to 1909, although the presence of ore in the mass of the hill itself had long been suspected, systematic exploration had not been attempted. A few shallow pits on the surface here and there had disclosed some chalcocite with associated carbonates as stringers in a mass of strongly brecciated, silicified, and iron-stained porphyry. The workings in the surrounding limestones had penetrated the porphyry in a dozen or so places for some distance from the contact but had disclosed no ore. However, the rock had been largely altered to quartz and sericite and very thoroughly impregnated with pyrite and traces of the copper sulphides. A close geological study of the conditions, both surface and underground, strengthened these suspicions, and plans for prospecting the hill were made in 1909.

Reference to the accompanying illustrations, Figs. 1 and 2, will give an idea of the general appearance of the surface. The mineralization is more intense in two main zones, one at the west end and the other at the east end of the hill.

The outcrop of the hill is elliptical and approximately 3,000 ft. long by 1,400 ft. wide.

The Sacramento shaft, which is the main hoisting shaft of the mine and situated at the east end of the hill, as shown by the accompanying map (Fig. 3), offered a convenient base for prospecting that end of the hill, while the workings from the Holbrook shaft offered a similar opportunity at the west end. The choice of level for the first work was determined by analogy with the most favorable depth for enrichment below surface observed in the other disseminated deposits of the Southwest.

* Chief Geologist, Copper Queen Consolidated Mining Co.



FIG. 1.—WEST END OF SACRAMENTO HILL FROM A POINT NEAR THE HOLBROOK SHAFT.



FIG. 2.—SOUTHEAST SIDE OF SACRAMENTO HILL, SHOWING THE SACRAMENTO SHAFT IN THE CENTER, THE TEST MILL AT THE RIGHT, AND THE POWER HOUSE AT THE LEFT.

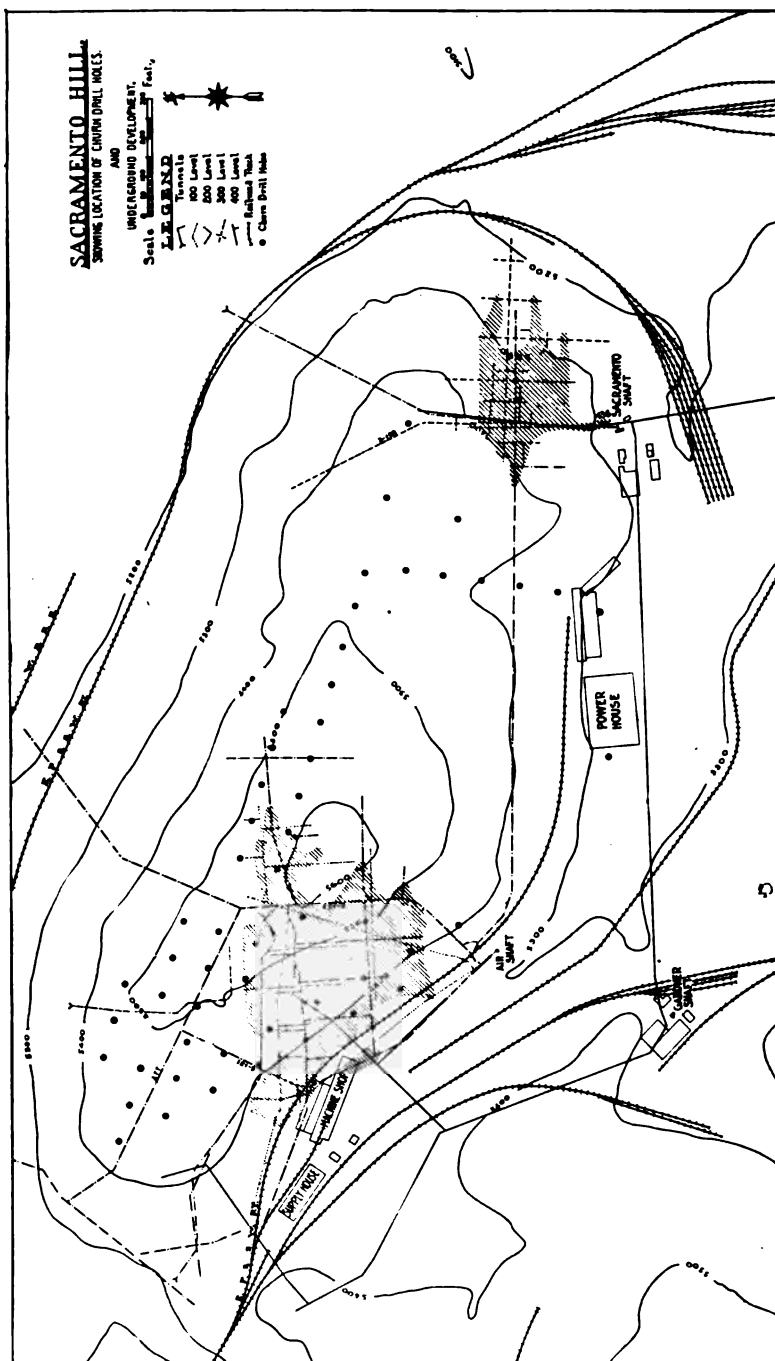


FIG. 3.—MAP SHOWING LOCATION OF CHURN-DRILL HOLES AND UNDERGROUND DEVELOPMENT ON SACRAMENTO HILL.

Accordingly, drifts on the 400-ft. level of the Sacramento shaft were started north and south (436 drift—see map) to crosscut the most favorable surface showings. This work disclosed about 90 ft. of mineralized porphyry, containing disseminated pyrite, chalcopyrite, and chalcocite, which averaged 2.0 per cent. copper by careful channel sampling. At the same time, 277 drift from the Holbrook workings was started east under the long axis of the hill to serve as a base from which to crosscut the best surface showings at the west end of the hill. This drift failed to disclose any commercial mineralization in the porphyry. Following this work, a tunnel was driven in from the north side of the hill, over 436 drift, which disclosed oxidized and leached porphyry with the exception of about 40 ft. of enriched material, averaging slightly under 2 per cent. A station was then cut in the Sacramento shaft, halfway between the tunnel level and the 400-ft. level, and a drift, 3-158, run north, which resulted in the exposure of 210 ft. of material averaging 2.21 per cent. copper. A crosscut from 277 drift, 2-131 drift (see map), was started and encountered 25 ft. of material averaging 3.8 per cent. copper.

The work of development was then undertaken in earnest, and up to February, 1913, when underground work was discontinued, a considerable tonnage, averaging 3.53 per cent. copper, was estimated to have been developed by 19,604 ft. of drifts, crosscuts, and raises.

At this time it was decided to continue the development by means of churn drilling. Both the lower cost of development and the greater desirability of the vertical sections of the ore obtained by the drills, as compared with the horizontal sections obtained with underground methods, were factors influencing this decision. The company had been operating a Star drill on somewhat similar ground. The costs obtained, exclusive of the cost of the drill, were as follows, for 8,000 ft. of drilling:

	Cost per Foot
Labor and supplies.....	\$1.99
Roads.....	0.71
Total.....	\$2.70

The general plan adopted for the drilling of Sacramento Hill was outlined as follows:

1. The drilling of a line of holes to crosscut the area of the hill between the two well-mineralized areas at either end. The purpose of this work was to determine at the earliest moment the approximate extent of the orebodies.

2. *a.* In case the orebodies were proved continuous by (1), the whole area of the hill was to be drilled at the corners of 200-ft. squares, with

intermediate holes where the irregularity in outline of the ore seemed to require it.

b. In case this work proved the central portion of the hill to be barren, the work was to be laid out on 100-ft. squares, with the ore developed underground as starting points. Here, too, when it seemed necessary, intermediate holes were to be driven.

With these objects in view, a road was constructed to drill the line of holes called for by (1). These holes showed no ore, and consequently the plan described under (2-b) was adopted and the roads laid out on the ground to cover the orebodies and their probable extensions with 100-ft. squares, with the minimum amount of road building.

The cost of the road building for the period considered was as follows:

Labor and supplies.....	\$22,065.77
Footage of road.....	15,092
Cost per foot.....	\$1.02

About this time, owing to the expense of delivery of the coal, and the proximity to the central power house (see map), it was decided that some cheaper power could be secured. The condensation losses eliminated the question of piping steam from the power house. Compressed air was barred out by the fact that the pressure of 100 lb. to the square inch at the power house seemed too low for the purpose. Delivery of fuel oil to the drill by pumping from the storage tanks at the power house was favorably considered, but finally electrical power was decided upon as offering the most advantages and the fewest disadvantages. Accordingly, two No. 22 Armstrong drilling rigs without engines or boilers were secured (see Fig. 4) and equipped with motors of the following specifications: 25 hp., 230 volts, 94 amperes, 400 to 1,200 rev. per minute.

At the same time, an arrangement for steering the machines when moving was devised, and installed on each machine. The moving was accomplished by means of "dead men" and the drilling rope and reel. This arrangement was somewhat less convenient than that of the steam drill, which was self-propelling. A transmission line 2,860 ft. long, of the following specifications, was installed from the power house at the following cost:

2,860 ft. cable, 0000 rubber-covered standard.

1,500 ft. cable, 00 flat mining machine (cut into 250-ft. lengths).

Labor.....	\$486.69
Supplies.....	1,434.74
Total.....	<u>\$1,921.43</u>

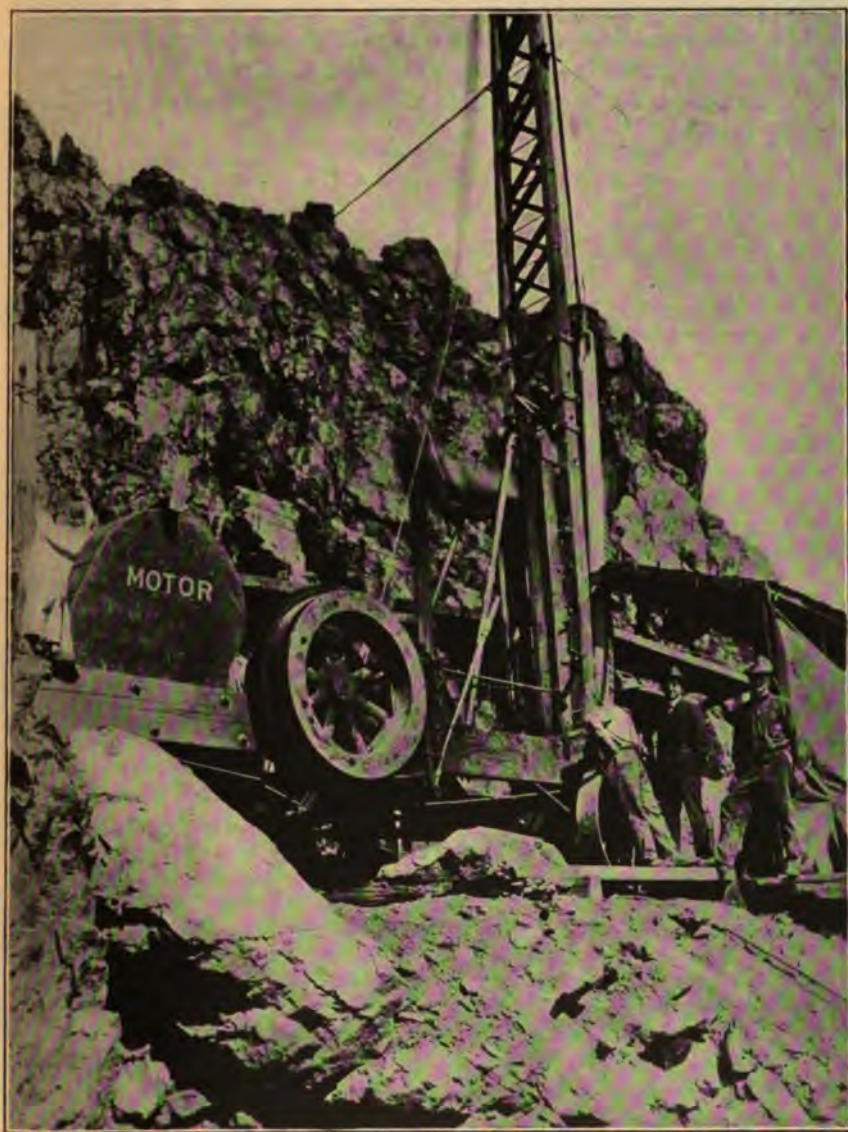


FIG. 4.—SIDE VIEW OF ONE OF THE ELECTRIC DRILLS IN OPERATION.

From the main line, which was run approximately parallel to the axis of the hill, temporary lines of the flat cable were run to the drill locations.

Tables I and II give the costs obtained in this work. The figures for No. 1, or the steam drill, cover a period of 22 months, during which time 12,153 ft. were drilled in outside ground, and 16,617.5 ft. in Sacramento Hill. The figures for the Nos. 2 and 3 electric drills cover a period of only $3\frac{1}{2}$ months.

TABLE I.—No. 1 Churn Drill (Operated by Steam)

	Cost	Cost per Foot	Per Cent. of Cost.
Labor:			
Drilling.....	\$13,497.62	\$0.469	33.5
Casing.....	2,113.50	0.073	5.2
Fishing.....	564.86	0.020	1.4
Moving.....	5,679.03	0.197	14.1
Repairing.....	3,488.94	0.121	
	\$25,343.95	\$0.881	62.8
Supplies:			
Coal.....	3,819.22	0.133	8.7
Casing.....	1,336.16	0.046	3.3
Drilling tools.....	2,880.33	0.100	7.1
Cordage.....	2,334.01	0.081	5.8
Miscellaneous.....	1,779.90	0.062	4.4
	12,149.62	0.422	30.1
Storehouse.....	126.11		
Water line.....	413.57		
Air line.....	16.95		
Steam line.....	275.20		
	831.83	0.029	2.1
Teaming.....	2,000.00	0.000	5.0
Total.....	\$40,325.40	\$1.402	100.0
Footage drilled, 28,770 $\frac{1}{2}$			
Footage drilled per 8-hr. shift,			
▶ 16.7	3,734.87		
Initial costs.....	\$44,060.27		

TABLE II.—Nos. 2 and 3 Churn Drills (Operated by Electricity)

	Cost	Cost per Foot	Per Cent. of Cost
Labor:			
Drilling.....	\$3,867.45	\$0.466	32.7
Casing.....	1,215.25	0.146	10.3
Fishing.....	597.24	0.072	5.0
Moving.....	1,696.82	0.204	14.3
Repairing.....	698.06	0.084	5.9
	\$8,074.82	\$0.973	68.3
Supplies:			
Casing.....	170.39	0.021	1.4
Drilling tools.....	548.56	0.066	4.6
Cordage.....	770.40	0.093	6.5
Miscellaneous.....	1,610.81	0.194	13.6
	3,100.16	0.373	26.2
Electric power.....	276.22	0.033	2.3
Teaming.....	370.65	0.045	3.1
Total.....	\$11,821.85	\$1.425	100.0
Footage drilled, 8,294			
Footage drilled per 8-hr. shift, 15.2			
Initial costs.....	6,578.75		
Electric power line:			
Labor.....	486.69		
Supplies.....	1,434.77	1,921.46	
Grand total drills Nos. 2 and 3..	\$20,322.06		
Grand total drill No. 1.....	44,060.27		
Assaying cost.....	1,132.59		
Roads:			
Labor.....	21,055.55		
Supplies.....	1,010.22	22,065.77	
Grand total.....	\$88,412.41	\$2.385	
Total footage, 37,064½			
Average depth of hole, feet, 427			
Footage of roads, 21,814			
Cost per foot of roads.....	\$1.01		

The following table gives the cost for the steam drill on Sacramento Hill alone:

TABLE III.—No. 1 Churn Drill (Sacramento Hill)

	Cost		Cost per Foot	Per cent. of Cost
Labor:				
Drilling.....	\$7,696.85		\$0.463	41.5
Casing.....	1,261.84		0.076	6.8
Fishing.....	253.51		0.015	1.3
Moving.....	2,939.66		0.178	16.0
Repairing.....	1,732.93		0.104	9.3
		\$13,904.79	\$0.836	74.9
Supplies:				
Coal.....		1,702.63	0.102	9.1
Miscellaneous.....		1,867.51	0.112	10.1
Water lines.....		237.39	0.014	1.3
Teaming.....		850.00	0.051	4.6
Total.....		\$18,562.32	\$1.115	100.0
Footage drilled, 16,617½				
Footage drilled per 8-hr. shift, 20.4				
Teaming estimated.				

It is not quite fair to compare the figures in Table III with those for the electric drills, as the larger part of the work by No. 1 was done in nearly barren ground of uniform character, while that of the electric drills was done entirely in ore-bearing ground, where the changes from strongly silicified leached capping to soft enriched ore, and harder primary material, rendered the drilling much more difficult. For a three months' period, in which the drills were working under essentially similar conditions, the following figures were obtained:

Operating Expense

	No. 1 Steam	No. 2 Electric	No. 3 Electric
Cost per foot.....	\$1.34	\$1.56	\$1.15
Footage drilled.....	4,169	3,198	4,510

Analysis of the working conditions shows that No. 2 drill was operating in exceptionally bad ground and that, while not conclusive, these figures indicate that eventually the electric drills should be operated at from 10c to 25c. per foot cheaper than the steam drill, under the conditions obtaining.

The work was conducted as follows: It was carried out under the immediate supervision of a drill superintendent, and operations were continuous in three 8-hr. shifts, excepting on Sundays. Each shift on each machine consisted of one driller, one helper, and one sampler. There was also a chief sampler, who was responsible for the sampling on

all drills. On day shift, there was one team with a driver and helper for approximately half a shift to deliver coal for the steam drill, wood for drying samples, to haul supplies, repair parts to and from the machine shop, casing, samples, and miscellaneous equipment when the drill was moving from one hole to another. The electric drills were equipped with portable forges operated by electric motors, while the steam drill was equipped with a forge operated by a small steam turbine. Both smithing coal and coke were used in these forges.

The routine duties of the various members of the crew were as follows: The driller was in charge of and operated the machine; the helper oiled the machine, dressed bits, and assisted and relieved the driller when necessary; the sampler arranged for the sampling of the sludge every 5 ft., the drying of the sample, the tagging and recording of the sample, and its transportation to the assay office. When not engaged in sampling, he was at the call of the driller for repairing, bit setting, or moving. The chief sampler, one-half of whose time was devoted to the supervision of underground sampling, had general charge of the drill sampling, and kept the office records up to date.

In the costs given, the sampler's time is included in the labor charge, as only a small portion of his time is really chargeable to sampling.

The tools used were standard and are of no particular interest in a discussion of costs, excepting that lack of proper tools, or a sufficient number of them, is undoubtedly a most frequent cause of high labor costs in churn drilling. The labor and supply charges for casing depend entirely on the character of the ground drilled and so comparisons of these items for the two kinds of machines are of no value. The time lost in fishing depends on the character of the ground, the supply of tools, and the nature of the power. Experienced drillers operating a steam drill are able to tell immediately when there is danger of the tools sticking by the character of the exhaust, and by taking necessary precautions can avoid undue strain on the drilling rope, and possible loss of the tools. In the case of the motor-driven drill, this becomes a matter of much more delicate observation, and to inexperienced men offers considerable difficulty. This is indicated by the increased loss of time in fishing with the electric over the steam drills, and the slightly increased cordage expense.

The labor of moving is practically the same with both types of drill. The repair labor item for the steam drill is higher, due partly to inherent disadvantages and partly to the fact that it was a second-hand machine.

In considering the cost of power on the two types we encounter the widest difference in the cost of operation. Unfortunately, the cost of teaming on the steam drill, at least 60 per cent. of which is chargeable to the coal, is not very accurate, owing to the fact that stable expense in the past has been very roughly segregated. This would indicate a cost of



FIG. 5.—THREE ARMSTRONG ELECTRICALLY OPERATED CHURN DRILLS ON THE WEST END OREBODY. THIS VIEW SHOWS THE CHARACTER OF THE ROAD BUILDING.

17½c. per foot for the coal and another 2½c. for pipe lines, or a total of 20c. per foot.

Electric power cost only 3.3c. There is no teaming to be charged against it, while the proper charge for the installation of the power line depends on the amount of drilling to be done. For a minimum of 30,000 ft. this would amount to 6.4c. per foot, or a total of 9.7c. per foot, or less than one-half the power cost of the steam drill.

The 4c. difference in the item of drilling tools in favor of the electric drill is more apparent than real, inasmuch as the steam machine was put into commission more than a year prior to the starting of the electric drills and some of the tools then purchased have been used on all machines.

The 13c. difference in miscellaneous supplies in favor of the steam drill, on the other hand, is due to the fact that a considerable portion of this item might very reasonably have been charged to initial costs.

The initial costs include the purchase price of the machines, the freight, and, with the exception just noted, all additional permanent equipment.

The road building completed is sufficient for a total of 60,000 ft. of drilling to an average depth of 400 ft., or a cost of 36.8c. per foot of hole drilled.

The assaying cost amounts to 3.5c. per foot.

The roads built are 9 ft. wide, excepting at turns, where they are 12 ft. wide. The maximum grade, of which there are two short stretches, is 30 per cent. There was a great deal of rock work and dry walling to be done of a character shown in Fig. 5. In general, the roads follow the contours of the hill.

The initial costs of the three machines, distributed over a minimum of 60,000 ft., amount to 17.2c. per foot.

These figures would indicate the following complete costs for 60,000 ft. of drilling:

	Cost per Foot
Initial costs.....	\$0.17
Operating expense.....	1.43
Road building.....	0.37
	\$1.97

A salvage of 7c. a foot on the initial costs might be reasonably allowed and it seems probable that the operating expense can be lowered another 5c. to 10c. per foot, giving an average cost of about \$1.80 per foot for a minimum of 60,000 ft. of drilling.

The checking of the churn-drill sampling, which after all is the most important part of the operation, has not progressed very far as yet. One raise has been driven on a churn-drill hole in ore with the following results:

Assay of Samples

	Churn Drill, Per Cent. Copper	Channel, Per Cent. Copper
	3.66	4.00
	3.72	2.50
	4.18	3.40
	2.38	5.60
	5.78	4.50
	4.25	3.70
	2.82	4.70
	3.30	2.80
		2.90
Total.....	34.21	34.10
Average.....	3.80	3.79

The channel samples were taken in 5-ft. lengths, about $2\frac{1}{2}$ ft. from the drill hole. The weight of sample taken was about 8 lb. per foot. The elevation of the top of the ore, as shown by the drill-hole record, was checked exactly by the raise; the hole, however, continued in ore below the level from which the raise started, so that the elevation of the bottom of the ore has not yet been checked. The results indicated are highly satisfactory.

The office records of the samples consist of longitudinal sections and cross-sections through the drill holes, showing the assay value of each 5-ft. sample, on which the probable outlines of the orebody are drawn, using all information available from both underground development and churn drilling. From these sections, the tonnage developed is calculated.

The conclusions warranted by the costs obtained so far, are

1. That 60 to 75 per cent. of the operating cost is labor.
2. That the labor item is materially affected by the supply of tools, and the accessibility to a machine shop for heavy repairs.
3. That the choice of power in churn drilling cannot affect the costs obtained, excepting to a very minor extent. In the present case, the low cost of electrical power, due to the proximity to the central power house, determined the selection.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Salida Smelter

BY F. D. WEEKS, * NEW YORK, N. Y.

THE Salida smelting plant, owned by the Ohio & Colorado Smelting & Refining Co., is situated at an altitude of 7,000 ft., about 2 miles west of Salida, Colo., and 215 miles southwest of Denver. Salida is a division point on the Denver & Rio Grande R. R. where the narrow-gauge line serving the San Juan region joins the through standard gauge. This narrow-gauge road also serves the coal and coke companies at Crested Butte and the lime quarries and mines at Monarch and Garfield, which are only 20 miles distant. These facts, together with the possibility of an appropriate site, doubtless influenced the builders to locate at this point.

The smelting plant and yard occupy 80 acres of land on the northeast side of the Arkansas River. The slope of the ground is such as to permit of the ore being delivered by cars at a point 25 ft. above the feed floor; and in front of the furnaces there is a fall of 50 ft. for the slag dump. A recent purchase of river bottom has provided slag room for 20 years.

As originally constructed, the plant consisted of four blast furnaces, 40 by 144 in. at tuyères, two copper-matting furnaces, dust chamber, stack, power plant, blowers, sampling works, and one railroad trestle about 16 ft. from bed floor to rails. Later 12 hand roasters of excellent design were added and the practice of smelting leady matte in a copper-matting furnace was abandoned. Since the plant passed under the present management the equipment has been increased so that the following is a brief description of the same at this date: Two lead blast furnaces, 48 by 180 in. at tuyères, 17 ft. 6 in. tuyères to feed floor; four lead blast furnaces, 40 by 144 in. at tuyères, 17 ft. 6 in. tuyères to feed floor; two copper-matting furnaces, not used; blast-furnace flue, 900 ft.; area 245 sq. ft.; blast-furnace stack, 150 ft.; above furnace floor, 190 ft.

When smelting a 500-ton charge per day, the gases approximate 63,700 cu. ft. per minute and travel at a speed of 260 ft. per minute.

The slag is hauled by a 10-ton electric locomotive; the slag shells and settler refuse are elevated to a bin on the feed floor by an electric hoist. The matte is tapped in 1,600-lb. cakes and loaded into cars

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without breaking, in the case of matte low in copper; copper matte is broken by hand before going to the sampler.

The bullion is tapped into pots holding 10 bars and poured into drossing kettles of 30 tons capacity, then drossed, sampled, and shipped. The furnaces, which are fed by hand, are not entirely satisfactory, because the brick walls are so thick that the feeders cannot get close to their work; and further, the furnaces have cast-iron jackets with an end bosh. These faults have not been corrected, for the reason that it would require rebuilding the furnaces, and lead smelters are not at present incurring any expense that can be avoided.

The roasting plant at present consists of three Godfrey furnaces, 26 ft. in diameter, with a capacity of 25 tons per day each; one Wedge furnace, 21 ft. in diameter, with a capacity of 75 tons per day; four Dwight-Lloyd machines, 42 by 264 in., with a capacity of 80 tons per day each. The Godfreys and Wedge are used for preliminary roasting before sintering on the Dwight-Lloyd machines, the sulphur in high-sulphur ores being reduced to a point where the product can be added to the Dwight-Lloyd charge without raising the sulphur above 15 per cent.

These furnaces are served by a 468-ft. flue, having a cross-sectional area of 166 sq. ft., and joined to a stack 85 ft. high, 12 ft. in diameter.

The volume of gases discharged is approximately 125,000 cu. ft. per minute.

The product from the Godfreys and Wedge falls in the center of a revolving cast-iron table and is moved toward the edge and finally scraped off by fixed plows, being moistened during its passage across the table by a spray of water. The result is very satisfactory, since there is no dust nor smoke and the product is in just the right condition as to moisture, being neither too wet nor too dry.

The Dwight-Lloyd product falls into side-dump railroad cars and is taken to smelting beds. Local conditions necessitate frequent changes of charge, so that the sinter is spread on smelting beds instead of being smelted separately.

This plant was one of the first to adopt the Dwight-Lloyd sintering machine, and while the management went through the usual experience that falls to the lot of beginners with a new process, it also had the pleasure of seeing the blast-furnace capacity increase, in the case of the 48 by 180 in. furnaces, from 180 to 250 tons per day for long periods.

Ore as received is handled as at all custom lead smelters, the only variation being that rather more screening is done in the case of oxidized ores, so as to send the coarse material to the blast furnace and the fine to the sintering machines.

While steam is kept up in one boiler for heating purposes and for some small machines, most of the power is purchased from the local

electric light and power company, which has a hydro-generating plant on the Little Arkansas River, a few miles above the town.

Water for boiler and jacket use is obtained from the Arkansas River, when it is clear; otherwise, from springs. The river water is better, carrying only 8.77 grains of solids per U. S. gallon against as much as 32.47 grains per gallon in some spring water. These springs are seepage springs caused by irrigating water leaching through the surface soil on the upper *mesa*. When spring water is used, the jackets must be washed once in 30 days.

The bare description of the plant with the dimensions of apparatus and appliances cannot be very instructive to beginners nor interesting for those who have been long in the work. However, it seems that some of the peculiarities of the Salida smelter as originally built should serve to prevent the younger men from making similar mistakes; the older men can get their satisfaction by reflecting that they never did anything quite so far removed from good practice.

Originally, a brick wall, with a narrow door in front of each blast furnace, was constructed. How the work was to be done behind this wall is not known, but it shows that the designer did not know how to do the work.

The water jackets were held in place by a 12-in. I-beam running the length of the furnace, thus effectually preventing any one from examining a tuyère or jacket.

The shaft walls on the feed floor were so thick that the feeders were shoving ore into one end of a tunnel, and, since the opening was not as wide as the shaft, the ends of the furnace could not be fed except by throwing the charge around a corner—which, of course, was not done.

The proposal to concentrate matte, carrying 20 per cent. lead and 5 per cent. copper, in a copper-matting furnace was intended to do away with the necessity of roasting equipment.

The largest Corliss engine on the plant was the exhibition engine at one of our world's fairs held some years ago. When repairs were needed, it was discovered that this was the only engine of the kind ever built, and the drawings of it were lost; so each repair part called for an accurate drawing.

In the track plan of the works, all the ore, coke, and limestone had to be unloaded from one track. This track also held the cars which were to be unloaded at the oxide crusher as well as at the sulphide mill. The problem of dispersing some of this congestion was not simple. It was solved by building two trestles in place of one; by having the limestone brought in dump cars so that unloading took little time; and by building an oxide crushing and screening plant on new tracks.

Originally the bullion was sampled by taking a dip sample from the lead well, which was not satisfactory. The present management built

drossing kettles in which the bullion is stirred by compressed air, skimmed into a Howard press, stirred by hand and siphoned out into bars for shipment, after sampling by dipping "gum-drops," approximating $\frac{1}{2}$ assay ton in weight. For some time the results were unsatisfactory. Although the copper had been reduced to 0.06 per cent., the bullion had a dirty appearance, and "gum-drops" of the same dip would not check with each other, especially in gold content. Consultation with men versed in the subject did not help. A remedy was finally found when the temperatures were determined by a pyrometer and the difficulties overcome by the following procedure: Take off heavy dross at 900° F.; blow until temperature reaches 680° F., skimming from time to time; raise the heat to 720° F. for sampling, and mold at 800° F. Since this method was adopted, little trouble has been encountered.

A plan to eliminate the personal equation in bullion assaying has been carried out for some years, with good results. The "gum-drops," weighing about $\frac{1}{2}$ assay ton each, are weighed without trimming, cupelled, and parted; the weights of the "gum-drops" in decimals of an assay ton, of the gold and silver together and of the gold alone, in milligrams, are reported by the assayer. The calculations are then made in the office on a machine. In this way the assayer is not tempted to make his results check, because until they are figured he does not know whether they will check or not. The excellent results obtained are sufficient reason for following the plan.

Referring again to the Dwight-Lloyd sintering plant, it must be borne in mind that this plant was built when there were no plants to serve as guides. The original charge-measuring and mixing apparatus is still used, although many accessories have been evolved since the construction of the plant. The charge is made up from six bins with belt bottoms discharging on to a belt that delivers the charge to the center of a cast-iron revolving table. The charge is moved to the edge of the table by fixed plows, water being added during the time of plowing; then it is scraped off into ore cars, sent to the top of the building on a platform elevator, and dumped into the machine feed bins. The use of a platform elevator instead of a bucket-and-belt elevator was decided on after seeing the segregation that took place when a belt elevator or conveyor was used. This stopped the segregation of coarse charge along the edge of the pallets.

For a long time coarse charge was fed to the pallets by means of an extra hopper behind the regular feed hopper, but it was found that if the back of the regular hopper was removed and the feed regulated to a thin stream, all the coarser particles would roll to the grate and stay there, stopping grate leakage and leaving only fines on top, presenting a proper surface for ignition. Coal ignition is still used because gas is not available and with a wooden structure gasoline is feared.

An important improvement has been made in the construction of the furnaces, by placing an adjustable rail by the side of the pallets so that the weight of the loaded pallets is carried on the rollers and not dragged across the top of the wind box causing excessive wear. The height of the rails is adjusted so that as the top of the wind box wears the rails take the loads. This arrangement leaves no appreciable space between the bottom of the pallets and the top of the wind box.

Many small improvements are planned that will effect savings without great outlay of money. Like all custom lead smelting works, the Salida plant at times is called on to handle more zinc than is comfortable. The chief lead supply is that from the Cœur d'Alène district of Idaho. Ores with iron in excess are obtained from Leadville, and the silica supply is helped out by receipts of clinkered zinc residue from plants of an allied company operating in Oklahoma.

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Some Suggestions Regarding the Determination of the Properties of Steel

BY A. N. MITINSKY,* PETROGRAD, RUSSIA

EDITED BY LAWFORD H. FRY

(San Francisco Meeting, September, 1915)

THE theory of elasticity, the science of the strength of materials, and all our calculations regarding engineering structures are based on Hooke's law, that in loaded bodies the deformations are proportional to the load producing them. Without this law it would be impossible to design intelligently engineering structures such as bridges, frames, axles, etc. This being the case, the proportional limit,¹ that is, the point at which deformation ceases to be proportional to load, is the most important factor in engineering design. Further, the working fiber stress should be fixed in reference to the proportional limit instead of, as is usual, in reference to the ultimate strength. The incorrect use of the ultimate strength instead of the proportional limit in setting the working fiber stress is probably due to two reasons: (1) it is a more delicate operation to measure the proportional limit; (2) it is usually assumed, though incorrectly, that the proportional limit is determined by the ultimate strength.

In Germany the point at which permanent set first appears is often taken instead of the proportional limit. The two points will coincide if Gerstner's law, that elastic deformations are proportional to the loads producing them both before and after permanent set, is true; and the limit of proportionality and the elastic limit or point of permanent set will both be represented on a stress-strain diagram, by the point at which the line ceases to be a straight line and becomes curved or changes its inclination. It is to be further noted that the difference in definition between proportional limit and elastic limit introduces a difference in methods of determination in the testing laboratory. The German method of determining the elastic limit by alternately loading and unloading

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¹ EDITOR'S NOTE.—In the MS., Mr. Mitinsky speaks throughout of the "elastic limit," but he defines this as "the point at which the extensions cease to be proportional to the loads." Therefore according to American terminology he is referring to the "proportional limit" and this term has been used throughout instead of "elastic limit" as originally written.—L. H. F.

the test specimen is a lengthy process and time has in itself a very considerable influence on the results of physical tests.

In this way a difference in definition may lead to a difference in the determination of the point sought. The real danger from erroneous ideas regarding the material to be tested lies, however, not in this possible difference of measurement, but in the mistake, so common in Europe, of considering the yield point as the "practical elastic limit." This is only true in the sense that the yield point can be measured practically and simply, but is not true in the sense that a measurement of the yield point will give a correct idea of the elastic qualities of the metal tested. The value of the yield point has no relation to the elastic properties of the material. If the yield point stood in fairly close connection with the proportional limit we could use it in judging of the quality of the metal, but unfortunately this is not the case.

The stress-strain diagrams of steel can be divided into two groups, of which Figs. 1 and 2 are typical.

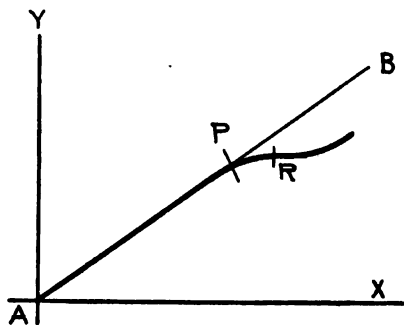


FIG. 1.

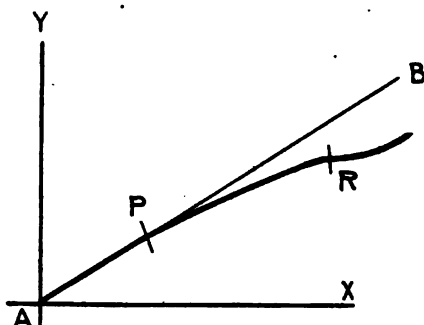


FIG. 2.

In these figures AB is the line representing proportionality between stress and strain, P is the proportional limit, and R the yield point.

Fig. 1 corresponds to a good steel which has been correctly treated both mechanically and thermally. This steel has the proportional limit and the yield point close together.

Fig. 2 represents a steel of poor quality. The proportional limit P is low, and beyond this point, P , the stress-strain line departs but little from the proportional line AB , until the yield point, R , is reached. The steel with a diagram of this character may show good elongation and reduction of area and a high ultimate strength, but is extremely undesirable for railroad service.

From a physical point of view, the proportional limit is the load at which the shearing stresses produced by a tension or compression load become sufficiently large to break down the metal. At the moment when this occurs, Hartmann's lines appear on the surface of the test

specimen, which becomes dulled, and at the same time the temperature rises. After a stress of this magnitude has been applied the metal is not in the same condition as before, its internal structure having been deformed.

By gradually loading and unloading a test specimen the proportional limit can be raised, and in many cases where the stress-strain diagram is of the type shown in Fig. 2, the increase is a substantial one. The proportional limit cannot, however, be carried above the yield point, and any blow or shock will reduce the artificially raised limit. It is further to be noted that this artificial raising of the proportional limit in tension is accompanied by a corresponding reduction in the proportional limit in compression, and inversely.

By subjecting metal to repeated and alternating tension and compression metal can be destroyed by so-called fatigue by loads very much below the ultimate strength. The resistance to fatigue depends solely on the proportional limit of the metal and is not due to some other special property. If the stress under repeated loads is less than the proportional limit no change occurs in the metal and no repetition of such a stress will cause injury, since the internal structure of the metal remains undisturbed. On the other hand, if the repeated stress exceeds the proportional limit the metal will be deformed and eventually ruptured. On the first application of the load producing such a stress, say in tension, a permanent deformation is produced, the shearing stresses do internal work, and the elastic limit in tension is increased. When the load is reversed on the following alternation the previously extended fibers are compressed and permanent deformation in compression is produced. This is the greater because the elastic limit in compression has been lowered by the previous deformation under tension. This process repeats itself, the metal being increasingly changed internally at each alternation.

In so far as fatigue of metal is concerned the proportional limit may be compared to the critical temperature in physics.

It would appear from the foregoing that the fatigue test, with stresses above the proportional limit, which is required by some English railways for tires and axles, could be replaced in practice by a determination of the proportional limit. It is sometimes said that a determination of the proportional limit is an impossibility in ordinary commercial testing, and doubtless the usual German method of making such determinations by applying and releasing the load has done much to justify this opinion. It is, however, a fact that in the Russian railway material inspection tests, determinations of the proportional limit are currently made and do not occupy more than 15 to 20 min. each. Marten's mirror apparatus and the Ewing and the Cambridge Scientific Co.'s extensometers are used. In this inspection testing it is not of importance to determine the exact value of the proportional limit, but rather to make sure that it is

not less than 20 or 25 kg. per square millimeter (28,400 or 35,500 lb. per square inch), as the case may be. In case the proportional limit is found to be less than the specified value the tires represented are rejected.

Having established the possibility of using the proportional limit in the specification, the Russian railways are now introducing it into their rail specifications, the feeling being that it is preferable to reduce the numbers of tests if necessary provided that the tests have a real significance.

If a bar of unstrained length l , Fig. 3, has a load gradually applied, the work absorbed while the elongation is increased from x to $x + dx$ is Sdx , where S_1 is the amount of load corresponding to the elongation x .

Writing W for the work absorbed by the deformation of the bar we have the equation

$$W_1 = S_1 dx \quad (1)$$

Now if w be the area of the bar, v its volume, so that $v = wl$, and E the modulus of elasticity

$$\frac{S_1}{w} = \frac{Ex}{l}$$

so that

$$W_1 = \frac{Ew}{l} x dx \quad (2)$$

If this be integrated for x between the limits 0 and λ , we get

$$W = \frac{Ew}{2l} \lambda^2 \quad (3)$$

where W is the work absorbed, or the internal work in the bar during its extension to the length $l + \lambda$. Again, if S be the load and Q the fiber stress corresponding to the extension λ

$$Q = \frac{S}{w} = \frac{E\lambda}{l}$$

and consequently

$$W = \frac{Q^2}{2E} v \quad (4)$$

Hence $\frac{Q^2}{2E}$ measures the internal work absorbed by the bar per unit of volume while being loaded to the stress Q . It therefore follows that for material with a proportional limit P the resistance to shocks and external work of any kind produced by loading is measured by $\frac{P^2}{2E}$, and that if a greater amount of work per unit of volume is to be absorbed a permanent deformation will take place. The modulus of elasticity being practically

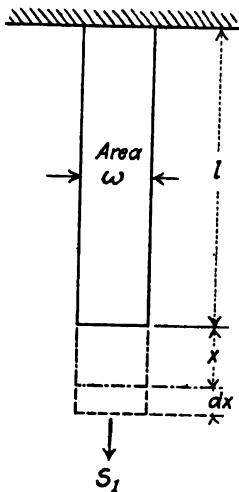


FIG 3.

the same for steels of the same analysis though differently treated (burning excepted), the resistance of the steel to blows, etc., is proportional to the square of the value of its proportional limit. For example, two pieces of tire steel both having 65 kg. per square millimeter (92,500 lb. per square inch) ultimate may have proportional limits of respectively 10 and 25 kg. per square millimeter (14,200 and 35,500 lb. per square inch) if the one is poorly and the other properly annealed. In this case the resistances of the two tires would be in the proportion of 1 to 6, while if it be remembered that the shrinkage which holds the tire on the wheel sets up an initial stress of say 8 kg. per square millimeter (11,300 lb. per square inch) the difference will be found to be still greater, say 1 to 70.

Ewing has proved that the elongation (l) or the compression produced by the shock of two bodies depends, not on the mass or dimensions of the bodies, but only on the relation between the velocity (v) of the body and on (c) the velocity of sound in the material under consideration; in other words, $l = \frac{v}{c}$. Now if the metal will be deformed if l is sufficient to produce a stress equal to P , the proportional limit, then under these conditions, with unit length $P = lE$ and combining this with the preceding equation, we have as a value for v_m the velocity of shock which will stress the body to the proportional limit

$$v_m = \frac{cP}{E} \quad (5)$$

Now in steel the velocity of sound (c) is about 4,900 m. (16,000 ft.) per second, and $E = 22,500$ kg. per square millimeter (32,000,000 lb. per square inch), from which it follows that to 10 kg. per square millimeter (14,200 lb. per square inch) of proportional limit there corresponds a dangerous velocity of shock of about 7 km. (4.4 miles) per hour. A good rail having a proportional limit of 25 to 30 kg. per square millimeter (35,500 to 42,700 lb. per square inch) will therefore have a dangerous velocity of shock of 18 to 21 km. (11 to 13 miles) per hour. According to Boussinesq the dangerous velocity for bending stresses is double that for tension. It is possible that the relation is not exactly as outlined above and that some coefficient should be introduced into equation (5), but it is evident that a rail or a tire with a high proportional limit is more likely to give satisfactory service than would one with a low value for this property.

The wear of rails and tires is of two kinds, (1) a crushing action producing a flow of the metal, and (2) an abrasive action. Wear of the first kind, or wear by deformation, depends, as has been just explained, only on the value of the proportional limit. Practical experience has shown that failure by deformation may occur in rails with high as well as with low tensile strength, but that it is in every case accompanied by a

low value of the proportional limit and that it is independent of the value of the yield point.

In the years from 1899 to 1906 a special committee was appointed by the Russian government to investigate the causes of rail wear and to work out means for securing satisfactory service from rails. This committee studied the question very thoroughly. A large number of rails made by manufacturers in Russia and other countries were subjected to chemical analysis and numerous physical tests, the rails being chosen so as to include those which had given the best as well as those having given the worst results in service. Table I sums up some of the results obtained:

TABLE I

Millions of Tons Carried by Rail	Number of Rails	Per Cent. of Carbon			Ultimate Tensile Strength, Pounds per Square Inch			Brinell Hardness Number		
		Min.	Max.	Mean	Min.	Max.	Mean	Min.	Max.	Mean
10 or less	46	0.165	0.615	0.411	72,500	118,000	92,500	163	233	194
10 to 50	72	0.127	0.705	0.342	65,400	118,000	88,200	144	246	191
Over 50	19	0.135	0.725	0.375	68,300	102,400	86,800	139	224	192

All tests for Russian railways are made using the International standard test specimen and the metric system.

In addition to other tests, drop tests were made on pieces of rail 5 ft. long, by dropping a $\frac{1}{2}$ -ton tup from a height h determined by the formula

$$h = 0.25 \frac{I}{z^2} \quad (6)$$

where I is the moment of inertia of the section and z the distance of the most strained fiber from the neutral axis. The permanent deflection after the first blow was noted in each case, the figures obtained being as follows: For good rails having given long service from 1.8 to 3.2 in., average 2.5 in., the best showing 2.6 in.; for rails with crushed heads 2.2 to 3.0 in., average 2.6 in.; for rails rapidly worn by abrasion 2.1 to 3.1 in., average 2.5 in. The committee was unable to establish any definite relation between the wear obtained from the rails in service and the ultimate strength, Brinell hardness, yield point, elongation, reduction of area, chemical analysis, or deflection on drop test. As a consequence little attention is now paid in Russia to ultimate strength and chemical analysis.

About four years ago over 20,000 tons of rails on the Siberian Railroad were found to be defective. These rails, which were all from the same steel works, showed crushed heads soon after having been laid. Table II herewith summarizes the results obtained by the Laboratory of Means of Communication at Petrograd. The first 12 lines give particulars ob-

tained from rails which had been in service, while the last seven lines refer to rails selected at the works. Several test pieces were taken from each rail and the maximum and minimum results are given in the table. It is evident that all 12 rails which crushed in service had low values for the proportional limit. In some cases it was only one-quarter the yield point. On further investigation it was found that the works, in the endeavor to secure a large output of rails from a weak mill, had finished them too hot.

TABLE II

C	Mn	Si	Ph	S.	Ultimate Strength	Elongation	Area Contraction	Yield Point	Elastic Limit	Brinell Number
0.55	0.35	0.011	0.036	0.01	65 to 67	11.5 to 17	27 to 35	38 to 45	9.5 to 9.5	158
0.55	0.22	0.007	0.069	0.006	68 to 73	13.5 to 19	27 to 37	27 to 39	9.5 to 10.5	161 to 174
0.45	0.44	0.034	0.043	0.007	62 to 64	13 to 14	44 to 46	38	16	150
0.63	0.39	0.013	0.037	0.005	71 to 71	14	24 to 28	28	9.5	180
0.54	0.42	0.012	0.055	0.029	63 to 66	15 to 19	31 to 42	40 to 47	10 to 14.5	156 to 167
0.37	0.22	0.009	0.051	0.020	58 to 61	11 to 20	44 to 50	35 to 46	16	154 to 158
0.38	0.44	0.012	0.085	0.019	59 to 63	13 to 44	43 to 50	43	16	158
0.48	0.66	0.018	0.035	0.025	66 to 68	13 to 19	37 to 40	45 to 50	14.5 to 16	158 to 167
0.42	0.86	0.040	0.087	0.008	61 to 68	11 to 20	51 to 55	41 to 55	16 to 22	156 to 176
0.56	0.59	0.012	0.046		78 to 80	8 to 15	11 to 24	45 to 47	13 to 16	187 to 194
0.48	0.57	0.012	0.031	0.018	65 to 70	13 to 18	26 to 40	42 to 49	10 to 11	160 to 176
0.54	0.35	0.018	0.036	0.023	63 to 68	19 to 20	35 to 40	35	9.5	153
0.69	0.44	0.028	0.050	0.029	85 to 87	9	16	49	12.5	194
0.65	0.73	0.17	0.083	0.021	73	17 to 18	31 to 36	41	19.5	168
0.61	0.18	0.09	0.045	0.029	75 to 76	14	21 to 25	41	20.5	174
0.50	0.36	0.03	0.035	0.021	68 to 69	17 to 18	29 to 36	45	16	163
0.63	0.25	0.017	0.053	0.021	79 to 80	13 to 14	26 to 27	47	20.5	185
0.55	0.39	0.09	0.043	0.038	73 to 75	15 to 16	22 to 32	34	16.5 to 24.5	174 to 180
0.57	0.36	0.11	0.060	0.039	80 to 81	11 to 14	21 to 25	37	29	194

The investigation of the influence of the proportional limit was continued by an examination of 12 tires from locomotives in main-line service on one of the Russian railroads. The tires were chosen from those removed from service after having reached the normal limit of wear, those showing the best mileage being selected. Three test pieces were cut from each, as near to the outer circumference as possible: Sample I from the flange, II from the center of the tread, and III from the face edge of the tread. The results are shown in Table III.

All of these tires which have given good results in service show a high value for the proportional limit, the average for all being 39,000 lb. per square inch.

If the rolling of a rail is finished at a high temperature the value of the proportional limit will be low, so that a measurement of the proportional limit will show whether or not the rolling was properly done. Again, tires will show a low proportional limit if improperly annealed; that is, if the critical temperature of say 875° C. has not been reached, or if the internal critical temperature has not been passed rapidly enough

TABLE III

Total run in thousands of versts ^a ...	363	337	337	329	289	284	273	238	171	165	164	152
Years of service....	11½	12½	10½	11½	12½	13½	7½	9½	6½	10½	6½	5½
Load per axle, tons..	13.3	13.3	13.3	13.3	13.3	13.3	14.4	14	12.8	14.4	14.4	13.3
Ultimate strength,												
I.....	70	71	69	64.5	70.5	70.5	67.5			75.5	57.5	69.5
II.....	63.5	74	67	62	67.5	70	69.5	73	69	78	52.5	65.5
III.....	68.5	74	70	67	68.5	71	71.5	77	64.5	83	55	76
Elongation,												
I.....	8	6.5	14	23.5	20	15.5	14.5			13	21	14.5
II.....	17.5	13	18	17	15.5	16.5	15	17	15	12.5	28.5	13.5
III.....	20	13.5	14.5	11.5	6.5	15.5	10.5	12	13.5	13.5	23.5	12.5
Area contraction,												
I.....		13	31	31	40	39	28			20	34	35
II.....	62	29	39	41	24	32	34	33	39	23	43	46
III.....	39	23	38	26	10	35	32	28	42	30	45	23
Elastic limit,												
I.....	29	27	26	23	33	28.5	18.5			21	22	29.5
II.....	22.5	25.5	19	27	30	31.5	29.5	28.5	27.5	32	17	31.5
III.....	31.5	25	26	35.5	32	25	22.5	31	27	40	20.5	25.5
Average.....	28	26	24	28.5	32	28	27	30	27	31	20	29
Carbon.....	0.46						0.50	0.56	0.47		0.42	0.46
Manganese.....	0.98						1.02	1.10	1.06		0.78	1.07
Silicon.....	0.09						0.121	0.197	0.181		0.191	0.292
Phosphorus.....	0.026						0.036	0.021	0.025		0.027	0.039
Sulphur.....	0.018						0.022	0.033	0.018		0.044	0.020

^a One verst is equal to 1.065 km.

during cooling. An unannealed tire rolled at too high a temperature will also show a low value for the proportional limit.

Generally speaking, a steel with a medium amount of carbon and having a fine regular internal structure will have a high value for the proportional limit. There are, however, two cases in which a high proportional limit does not correspond to a good steel. These are: (1) overheated steel, Widmanstedt's structure; (2) high phosphorus or arsenic, or too much manganese in combination with a comparatively high percentage of carbon. In both cases the metal with a high proportional limit may be brittle, but this condition will be disclosed by a drop test.

It may therefore be concluded that if a rail or a tire shows a high proportional limit and sustains satisfactorily a drop test, the steel is of as good a quality as can be desired, and no attention need be paid to the ultimate strength or Brinell hardness (these two are practically the same thing), nor to the elongation, reduction of area, carbon, manganese, etc.

The foregoing has dealt with wear of rails and tires by flow of metal. The wear by abrasion has not yet been studied as closely as it deserves. Practically all that can be said at present is that sulphur in steel tends to increase abrasion as it segregates sharply and may be present on the wearing surfaces in considerably higher quantities than in the average analysis of the ingot. Many persons are of the opinion that the ultimate strength has a great influence on the resistance of the steel to abrasion, but the author has been unable to find any sound basis for this opinion.

At the meeting of the International Society for Testing Materials in New York in 1912, Robin, Saniter, Rosenhain, and at Copenhagen in 1908, Musbaumer proved that this opinion is erroneous, and the same conclusions were reached by the Russian Rail Committee referred to above. The author therefore feels that the only honest answer to the question "On what does the abrasive wear of steel depend?" is "Except in the particular case of sulphur, we do not know."

It is sometimes said that the Brinell hardness number bears a relation to the abrasion, but the absence of such a relation was shown by the same tests as disproved the theory of a relation between abrasion and ultimate strength. The reason is clear. The scientific determination of hardness by Hertz in 1882, defines it as the value of the normal pressure per unit of area at the center of the bearing surface of a spherical body at the moment in which the elastic limit is attained. Auerbach was the first to carry practical tests beyond the elastic limit, and in all Brinell's experiments this limit is exceeded and permanent plastic deformations alone measured. It follows that Brinell's hardness numbers can be related only to the ultimate strength and are quite foreign to the true hardness and therefore to the wear by abrasion.

The author formulates the following suggestions regarding the quality of metal generally:

1. The proportional limit is the proper basis of all engineering calculations.
2. The proportional limit is independent of the yield point and of the tensile strength.
3. The proportional limit is the most important property, and should be as high as possible, especially for metals subjected to repeated loads, shocks, wear by crushing, etc.
4. The wear of metals of all kinds is independent of the ultimate strength, yield point, and Brinell number.
5. Wear by flow of metal or crushing depends only on the proportional limit. Wear by abrasion is not completely studied and it can only be said that sulphur has a deleterious influence.
6. The proportional limit can and should be measured in engineering inspection of materials.
7. The allowable working stress of any material should be fixed as a proportion of the proportional limit.
8. The resistance to fatigue is directly dependent on the proportional limit, and provided this is not exceeded no number of repeated stresses will destroy the metal.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Electro-Metallurgical Industries as Possible Consumers of Electric Power

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I. INTRODUCTION

THE utilization of hydro-electric power in electro-metallurgical industries, aside from purely mechanical operations, may be of two kinds. The electric energy may be used to supply the heat necessary for the performance of the chemical reactions of a metallurgical process, or the electrolytic action of a direct current may be employed for extraction of metals, either in a reduction process or a refining process. In discussing this subject, it is well to distinguish between established industries, which are producing metals and alloys at a profit by use of the electric current, and the many processes which are either still in the experimental stage or have not even passed the paper stage of development of a patent. That is, we consider electro-metallurgical industries as commercial operations which are now consuming large blocks of electric energy, while processes which are not in commercial operation are only prospective consumers. Also, owing to the difficulty and risk of any attempt to foresee the effect of the European war upon electro-metallurgical industries, we have felt obliged to disregard the war entirely, and state our opinions regardless of its existence.

Application of Electric Energy to Metallurgy

Disregarding electrolytic refining processes, the applications of electric energy to metallurgy, when considered from a commercial viewpoint, may be classified as follows:

Industries	Experimental Processes
Manufacture of aluminum	
Electric-furnace manufacture of:	Electric-furnace smelting of:
Ferro-alloys	Copper ore
Pig iron	Zinc ore.
Steel	

In this paper only the above industries and experimental processes will be considered as possible consumers of hydro-electric power in large quan-

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tities. Electrolytic refining processes are not included above, since they are either so closely interwoven into a cycle of metallurgical operations as one single step as to prohibit their operation as separate enterprises, or the electrolytic process is on such a small scale of operation as to be unimportant as a consumer of power. An example of the former is the electrolytic refining of copper. As at present operated, the electrolytic copper refinery, while it may be located at the copper smelter, is usually owned by a smelting company upon which it largely depends for its supply of unrefined anode copper. It has also been found advantageous to locate copper refineries near the market for refined copper, which is New York. An extreme illustration of a small-scale electrolytic operation which is not worth considering as a large consumer of power is the electrolytic refining of gold and silver.

All of the above electro-metallurgical industries are adaptable, under proper conditions, to self-contained development in a single plant. Practically none of these industries consume less than 1,000 hp. in a single installation. There are many electro-metallurgical processes in successful use for the extraction or refining of metals which do not consume large amounts of power, because the yearly yield is small, as in the case of gold and silver; or production is reduced by the small market, as, for example, for calcium, magnesium, or bismuth. If there were a demand for them, the two former metals might be produced profitably in conjunction with the manufacture of sodium, which, while a larger consumer of power than either calcium or magnesium, is not a large consumer in comparison with other electro-metallurgical industries.

There are many electro-metallurgical processes which might be considered as possible consumers of power, but in the majority of cases they exist only as patent specifications and have not yet been tested by either small or large experimental work. As an immediate market for power, most of these processes are of no interest to the power producer.

Factors Influencing the Success of an Electro-metallurgical Industry

The commercial success of an electric furnace or electrolytic industry, in which the basic process employed is technically sound, may depend largely upon the following factors: (1) the market for the product; (2) the cost of electric power; (3) freight rates; (4) labor conditions; (5) cost of raw materials.

Not only the cost of electric power, but all the other factors named, will influence, to a considerable degree, the success of any manufacturing project. If there is not a steady market for the product, permitting a profit on the investment, a low cost of power will not insure business success. There is but a limited market for many electro-metallurgical products, so that while a good profit might be shown on paper by comparing the cost of production with the average selling price, the enterprise

would be a failure with a large part of its products unsold. In the face of strong competition, success depends to a large extent, as in any other line of business, upon the efficiency of operation, quality of the product, and ability to meet competitive prices.

Reduction processes, as a general rule, require a very low power cost, especially those operations, producing a large tonnage of a comparatively cheap product, which compete with combustion processes, as, for example, the electric smelting of iron ore and zinc ore. On the other hand, electric-furnace refining processes do not require for commercial success an extremely low power cost. Many electric steel furnaces are operated at a profit on a power cost of 1c. per kilowatt-hour, or \$65.70 per horsepower-year, while few electric-furnace reduction processes can operate profitably with a power cost of over 0.3c. per kilowatt-hour, or \$20 per horsepower-year, and for complete assurance of commercial success, the power cost should be as small as from \$10 to \$20 per horsepower-year.

But, we repeat, the commercial success of an electro-metallurgical enterprise does not depend entirely upon the cost of power. Freight rates have a large influence on it. Generally speaking, in the western part of the United States, no such enterprise producing a large tonnage would have much chance of success unless located within a few hundred miles of the sea-coast, because of the high freight rates prevailing in the West as compared with Eastern rates. The short distance to water shipment has been a large factor in the success of Norwegian and Swedish plants, as well as of those in Switzerland and the French Alps. Practically all of their product is exported to foreign countries by water, while most of their ore and coal or coke is shipped to them by sea. With the exception of some specific raw material near by, the cost of raw material will depend largely upon freight rates, for in the majority of cases, at least the ores used must be brought from a distance.

II. PRESENT COMMERCIAL STATUS OF ELECTRO-METALLURGY

Aluminum

All aluminum manufactured to-day is produced by an electro-metallurgical process, either by the Heroult process or the Hall process, which are essentially the same. In the United States the Hall process is used and in Europe the Heroult process.

The production of aluminum in the United States has not increased materially since 1912 but exact figures are difficult to obtain, because the available statistics for recent years are those of consumption. The consumption in 1913 amounted to 72,379,090 lb., of which probably about 30,000,000 lb. was imported during the calendar year. Importations during the year ending June 30, 1913, amounted to 26,642,112 lb., and imports were increasing at the end of that period, but during the last quarter

of 1913 the consumption of aluminum decreased largely, in common with the reduction of business activity in other lines throughout the United States. The actual production of 1913 in the United States may be roughly estimated at 42,000,000 lb. In 1907, 60,000 hp. was being utilized in the manufacture of aluminum in the United States. To-day probably 80,000 hp. is in use for this purpose.

Until recently the production of aluminum in the United States was entirely controlled by the Aluminum Co. of America, owners of the Hall patents, which have now expired. In this country that company has one plant in operation at Niagara Falls, N. Y., and one at Massena, N. Y.; and in 1914 it started a new plant at Maryville, Tenn., where it is reported that 100,000 hp. can be developed. In Canada, it has a plant at Shawinigan Falls.

On the expiration of the Hall patents in the United States, a company was formed by French financiers interested in the Aluminium Française, under the name of the Southern Aluminium Co. A plant is in course of construction at Whitney, N. C., on the Yadkin River, where a total development in high water of 100,000 hp. is possible, although the average power will be about 45,000 hp. The capacity of the plant to be erected will be 20,000,000 lb. of aluminum per annum. This company continued its construction work up to the outbreak of the European war, when it became embarrassed financially, and was obliged to discontinue its work. The Aluminum Co. of America is still, therefore, the only producer of new metal in the United States, the remaining production being furnished by concerns that resmelt scrap and junk.

In Europe the aluminum industry has progressed as rapidly as in the United States. From the table given below it will be seen that Europe produced in 1913 about three times as much aluminum as the United States; but this country leads in individual production, with France second.

TABLE I.—*World's Production of Aluminum*
(From Statistical Report of the Metallgesellschaft, Frankfurt am Main)

	1911 Metric Tons	1912 Metric Tons	1913 Metric Tons
United States.....	18,000	19,500	22,500
Canada (exports).....	2,300	8,300	5,900
Germany.....			
Austria-Hungary.....	8,000	12,000	12,000
Switzerland.....			
France.....	10,000	13,000	18,000
England.....	5,000	7,500	7,500
Italy.....	800	800	800
Norway.....	900	1,500	1,500
Total	45,000	62,600	68,200

There are five large companies producing aluminum in France in connection with the manufacture of other electro-metallurgical or electro-chemical products, such as ferro-alloys, electric-furnace steel, calcium carbide, chlorates, sodium, potassium, and cyanides. Since all these companies manufacture numerous products, it is impossible to segregate the capital invested, or the power consumed, in making aluminum; but practically all of the French aluminum is made by these companies, which sell their product through a syndicate called Aluminium Française.

It is probable that there will be a large increase in the aluminum production of Norway very shortly, as one English company has recently completed a plant of 2,000 tons annual capacity at Vennessa, near Christianssand, and a Norwegian company has almost completed its plant near Arendal at Eydehaven, which when working full will take 25,000 hp.

Ferro-Alloys

The growth of the ferro-alloy industry in Europe has been rapid since 1899, but comparatively slow in the United States. There are about 25 European plants engaged in the manufacture of ferro-alloys by the electric-furnace method, as compared with two in the United States. There is, however, an electric-furnace ferro-silicon plant in Canada, at Welland, Ont.

There are several reasons why the growth of this industry has been slower in America than in Europe. Hydro-electric power is not so cheap here, and not so favorably located for the receipt of raw material and the sale of product. The water-power sites cannot be developed as cheaply as many of the foreign sites, where the cost of electric power per horsepower-year varies from \$7. to \$15 as compared with \$15 to \$30 in the United States, for power delivered at the manufacturing-plant transformers. In Canada, power is somewhat cheaper; but is often located in inaccessible places. Most of the Norwegian and Swedish plants are located at tidewater, or on navigable rivers. French works are within a couple of hundred miles of Marseilles. The use of ferro-alloys in the manufacture of high-class steels did not advance as rapidly in the United States as in Europe, and owing to less favorable natural conditions, electro-chemical and electro-metallurgical industries in general have not had so rapid a growth here.

A large proportion of the ferro-alloys used in the United States are imported, since, although there is a duty, local manufacturers do not supply the whole demand. This is true of about one-half of the ferro-manganese and one-half of the ferro-silicon used in the United States, as well as a large part of the ferro-tungsten. More ferro-titanium and ferro-vanadium are manufactured here than abroad. Our ferro-chrome production just about supplies the local demand.

In Europe the industry of manufacturing ferro-alloys in the electric furnace is in excellent condition commercially, with the demand for alloys steadily increasing. Because of the large navies built by European countries, there has been a great demand for ferro-chrome. The sale in Europe of ferro-alloys, especially ferro-silicon and ferro-chrome, is accomplished by the various plants combining in a syndicate, with each plant receiving a portion of the total market demand, according to arrangement.

In the United States, the Titanium Alloys Manufacturing Co. has a plant for the production of ferro-titanium at Niagara Falls, N. Y. The Electrometallurgical Co. has a plant at Kanawha Falls, W. Va., and another at Niagara Falls, N. Y. This company makes ferro-silicon, ferro-chrome, ferro-tungsten, ferro-vanadium, ferro-molybdenum, and ferro-phosphorus. The Primos Chemical Co., Primos, Pa., manufactures metals and ferro-alloys by chemical methods or in combustion furnaces. Among its products are ferro-tungsten, ferro-vanadium, tungsten metal, ferro-molybdenum, ferro-chrome, ferro-nickel, and ferro-boron. The American Vanadium Co., Bridgewater, Pa., manufactures ferro-vanadium by a method similar to the thermit process. The Goldschmidt Thermit Co. has a plant for the manufacture of metals and ferro-alloys by the thermit process at Newark, N. Y., but imports most of its products from its foreign works. This company produces ferro-titanium, ferro-vanadium, ferro-molybdenum, ferro-silicon, ferro-chrome, and chromium, as well as other metals and alloys. It has been recently reported that

TABLE II.—*Importations of Ferro-Alloys into the United States during the Year Ending June 30, 1913*

	Rate of Duty	Quantity Tons	Value	Value per Unit
Ferro-manganese	\$2.50 per ton	128,136.55	\$5,484,829	\$42.80
Ferro-silicon, over 15 per cent. silicon	20 per cent.	9,257.18	574,494	62.06
Chromium and ferro-chrome, value \$200 per ton or less. . .	25 per cent.	459.09	53,624	116.80
Value more than \$200 per ton	20 per cent.	68.29	35,667	63.55
Ferro-phosphorus, value \$200 per ton or less	25 per cent.	81.38	3,689	45.33
Molybdenum and ferro-molyb- denum, value more than \$200 per ton	20 per cent.	7.00	15,939	3,187.80
Titanium and ferro-titanium, value \$200 per ton or less. . .	25 per cent.	7.00	306	43.71
Value more than \$200 per ton	20 per cent.	19.21	9,213	479.59
Tungsten and ferro-tungsten, value more than \$200 per ton	20 per cent.	654.30	795,467	1,215.75

the Noble Electric Steel Co., Heroult, Cal., is manufacturing ferro-manganese at its electric-furnace plant.

Imports of ferro-alloys into the United States during the year ending June 30, 1913, are given in the table on the opposite page. The rates of duty were those in force previous to the passage of the present tariff, under which all duties were reduced considerably, in most cases down to 15 per cent. ad valorem. Hence it may reasonably be expected that imports will increase at some time in the future.

Pig Iron

The electric furnace for smelting iron ore is of advantage only in localities where charcoal and coke are expensive, and electric power is cheap. In the manufacture of pig iron, the electric furnace consumes one-third of the carbon used by the blast furnace, and hence its use may be advantageous where coking coal is scarce, and charcoal expensive. In considering the electric smelting of iron ores with regard to its commercial status at the present time, it must be remembered that by reason of the cheapness of water haulage, the electric furnace is in this case competing directly with the blast-furnace product, regardless of its location. The situation is not like that of the aluminum or ferro-alloy industry, in which the electric furnace has the field to itself, because of its technical and commercial superiority over any combustion process. Hence, in proportion to the amount of pig iron produced, we cannot expect to show nearly as large a rate of increase for the electric-furnace process as in the case of aluminum and ferro-alloys.

That the electric furnace has been successful in the smelting of iron ores in those districts which present favorable conditions is shown by the fact that ten furnaces of the Trollhättan type and one of the Helfenstein type, with a total power capacity of about 40,000 hp., are in operation in Sweden. In this country there is one electric-furnace pig-iron plant of two furnaces with a total capacity of about 7,000 hp., at Heroult, Cal. As previously remarked, the California furnaces are of the rectangular type. While the electric smelting of iron ore has been technically successful at Heroult, it does not appear to have been as profitable as in Sweden, by reason of the cost of reduction with charcoal or coke. Although the electric furnace uses only one-third as much coke or charcoal as the blast furnace, yet on the Pacific Coast of the United States all solid reducing agents are so scarce and expensive that this appears to be the great problem of electric, as well as of blast-furnace smelting of iron ores. Attempts to use oil have not yet proved successful.

While advances in electric smelting of iron ore have been satisfactory, considering that its field of use is limited, the actual tonnage capacity of electric pig-iron furnaces is small. The total capacity, in power con-

sumption, of the electric iron-smelting furnaces thus far erected is 47,000 hp., which would produce about the same amount of pig iron per day as one modern blast furnace of 450 tons output per 24 hours.

Steel

In 1904, only four electric furnaces were running in Europe for the manufacture of steel. To-day there are 114 electric furnaces producing steel in Europe and the United States, and 30 others are in course of construction. As in other electro-thermic processes, development has not been so rapid in the United States as in Europe. Only 14 furnaces are in this country. The average capacity per charge of the furnaces already built is 3.7 tons, whereas that of the furnaces under construction is 4.5 tons, an increase of 21.6 per cent. The total charge capacity of the furnaces now installed is about 250 tons per charge, and the total charge capacity of the furnaces under construction will be 170 tons per charge. The arc furnaces vary in capacity from 1 to 15 tons and require from 200 to 1,500 kw. for operation. A Heroult furnace of 25 tons capacity, requiring 3,000 kw., is nearly completed at Brückhausen, Germany. The induction furnaces vary in capacity from 1 to 10 tons and require from 165 to 600 kw. for operation.

Of the 114 furnaces in operation, 84 are arc furnaces and 30 are induction furnaces; of the 30 under construction, 26 are arc furnaces and 4 induction furnaces.

Table III gives the annual production of steel in electric furnaces, by countries, for the years 1908 to the first half of 1912.

From Table III it may be seen that for a new process in so conservative an industry as iron and steel manufacture, progress has been very rapid since 1908. Germany leads all countries in the steady growth of the process and the total tonnage produced. Although in Germany the production of electric-furnace steel increased 67.8 per cent. in 1911, in the United States it decreased 44.2 per cent. The decrease in this country was probably due to the conservatism of American steel makers, which has prevented the wide adoption of the process before experimental results had conclusively proved its merits. From present indications there will be a considerable increase in the production of electric-furnace steel in this country in the near future, although a very small tonnage, 6,882 tons, was reported by the American Iron and Steel Institute as the output for the first half of 1912. Of the total production of electric-furnace steel in the United States in 1911, 27,227 tons was ingots, and 1,878 tons castings. Of the total tonnage of electric-furnace steel made here in 1911, 6,700 tons was alloy steel, and 462 tons was rolled into rails. The large production of steel in the United States and Germany, in proportion to the number of furnaces operating, is due to the use of

TABLE III.—*Yearly Production of Electric-Furnace Steel*

Country	1908			1909			1910			1911			1912
	Produce- tion	Number of Fur- naces	Tons	Produce- tion	Number of Fur- naces	Change in Produce- tion	Produce- tion	Number of Fur- naces	Change in Produce- tion	Produce- tion	Number of Fur- naces	Change in Produce- tion	First Half
Germany and Luxemburg...	19,536	8	17,773		8	9.0	36,188	13	+104.1	60,654	15	+ 67.8	Tons
United States.....	55	1	13,762		4		52,141	7	+280.0	29,105	9	- 44.2	6,882
Austria-Hungary.....	4,333		9,048			109.2	20,028		+120.8	22,867	10	- 14.0	
France.....	2,686	7	6,515		12	143.0	13,445	21	+106.2	13,850	21	- 3.0	7,920
Sweden.....			591		11		431	12	- 27.0	2,034	13	+372.0	
Norway.....								1			1		
England.....											12		
Italy.....					5			2			4		
Switzerland.....											2		
Belgium.....											2		
Russia.....											1		
Total.....	26,610	16	47,039		40	78.2	120,116	56	+155.4	126,476	90	+ 5.22	

molten Bessemer and open-hearth steel, instead of cold scrap. The use of the latter almost entirely accounts for the comparatively small tonnage produced by France in proportion to the number of furnaces in operation. No figures were obtained for England, but it is probable that at least 10,000 tons of electric-furnace steel is manufactured annually in England. It is estimated that about 12 furnaces operate there, several of which receive hot-metal charges. Italy, Norway, Switzerland, Belgium, and Russia produce small tonnages also. The small increase in the total electric-furnace steel production for 1911 over that produced in 1910 was caused by the big decrease of production in the United States.

In the first years of its development, the electric-furnace process was considered as a competitor of the crucible process only, for making high-class steel from scrap iron and scrap steel; but with the successful operation of larger furnaces the electric process is likely to become an important adjunct to the Bessemer and open-hearth processes as a means of super-refining their molten products. The electric process, however, does not appear to be destined to supersede either of these methods, since greater efficiency and economy are obtained by a combination of any two of the three processes as a duplex process. The success of recent experiments has obtained for the electric process a definite place as a super-refining method. In time, preliminary refining will probably be done mainly in the Bessemer converter, the process being finished in the electric furnace, or the open hearth. In Europe, the electric-furnace process for making steel of the highest grades is rapidly superseding the old crucible method, because of its greater economy of operation and the possibility of using materials of lower grade.

Copper

So far as we are aware, no copper ores are treated in the electric furnace in this country at the present time. It is reported, however, that in Norway trial smeltings of copper ores with an electric furnace of 1,000 hp. and an estimated producing capacity of 2,000 tons of copper per annum have been conducted at the Ilen Smelting Works, Trondhjem, and we understand that it is the intention to smelt copper ores regularly at this plant in the electric furnace.

More or less experimental work has been done upon the subject, and, as a result of this work and reasoning by analogy, there seems no good reason why copper-bearing ores cannot be as successfully treated in an electric furnace as in a combustion furnace. In all furnaces of the latter class which are used for the treatment of copper ores, the fuel used takes no part in the reactions which are necessary for obtaining the desired product, unless it be in the reduction of oxide ores which are smelted alone—that is, without an admixture of sulphides—which is practically an unheard-of thing in this country at the present time. For example, in the

reverberatory furnace the fuel acts only as a heating agent; in ordinary blast-furnace smelting the coke added to the charge is for the purpose of supplying the heat necessary to raise the charge to such a temperature as will permit the necessary reactions between the oxides, sulphates, and sulphides present, and to scorify the resultant mass and thus permit the separation of the slag and matte. In semi-pyritic smelting the necessary oxidation of the sulphides and iron is brought about by the oxygen of the air entering the tuyères, and the coke used is simply for the purpose of supplying the amount of heat necessary for the successful carrying out of the process, which is not supplied by the oxidation of the sulphur and the iron present in the charge at the time it passes through the tuyère zone of the furnace. Such being the case, there seems to be no reason why the smelting of copper ores could not be done as well by electric heat as by that derived from the combustion of coke, especially if local conditions warrant it. The writers, as a result of experimental work which they have done in connection with tests which have been made by the United States Bureau of Mines, and likewise judging from results which have been obtained by others, believe that it is perfectly feasible, both metallurgically and commercially, under suitable local conditions, to use the electric furnace for the smelting of copper ores.

Zinc

Although, as previously remarked, more progress has been made hitherto in the electric smelting of zinc ores than in that of any of the non-ferrous metals except aluminum, and metals forming ferro-alloys, such as silicon, chromium, and tungsten, the process is nevertheless still largely in the experimental stage. There is no plant operating on a commercial scale except the Trollhättan works, taking from 10,000 to 13,000 hp. There are about 24 furnaces installed at this plant, each requiring from 400 to 1,200 hp. The same company, the Norse Power and the Smelting Syndicate, also has a smaller plant near Trollhättan at Sarpsberg, where there are seven small furnaces. One other small commercial plant is in course of erection at Keokuk, Iowa, by the Johnson Electric Smelting Co. It appears that the experiments conducted at Hartford, Conn., for several years have proved successful enough to warrant the installation of a small commercial unit to test the process further. The Johnson process and the Trollhättan process are essentially the same. Johnson claims to have overcome the problem of condensation of zinc vapor into zinc, instead of blue powder.

From the work at Trollhättan and the results of others, it is very evident that the difficulty in electric smelting of zinc ores lies almost entirely in the condensation of zinc vapor to a metal, rather than blue powder, under the peculiar conditions of the electric furnace. The

electric furnace presents no great difficulties, mechanically or electrically, because all the troubles formerly experienced have been solved in the construction of large pig-iron, steel, carbide and ferro-alloy furnaces. The problem, then, is one of a metallurgical nature, and is caused by the different conditions and greater speed of smelting in the electric furnace, as compared with the combustion retort.

While this problem is difficult, there is no reason why it should not be worked out in time. When it has thus been rendered unnecessary to resmelt a large proportion of blue powder (as at Trollhättan, where 2 tons of blue powder are smelted for each ton of ore treated), it is probable that electric zinc smelting will proceed rapidly in favorable localities. The use of iron as a desulphurizing agent does not seem to have advanced as far as the reduction of oxide with carbon, and it is probable that the latter will keep its present supremacy.

III. IMPORTANT ELECTRO-METALLURGICAL CENTERS AT THE PRESENT TIME

As we have already pointed out, the commercial success of an electro-metallurgical industry does not depend upon the cost of power alone but likewise on the market for the product, freight rates, labor conditions, cost of raw materials, etc. Each of these has to be considered in determining the feasibility of a new electro-chemical or metallurgical enterprise, and in this connection it is well to consider the factors which have made Norway and Sweden, France and Switzerland, New York harbor and Niagara Falls the four great electro-chemical centers of the world. We will briefly consider two of these places as regards: (1) the market for the product; (2) the cost of electric power; (3) freight rates; (4) labor conditions; (5) cost of raw material.

The Coast of Norway

The principal electro-chemical industry of Norway is the manufacture of nitrogen products, both nitrate and cyanamide, from atmospheric air.

1. Norway is remote from the larger centers of civilization.
2. It has the cheapest electrical power that is produced at the present time, for it is authoritatively stated¹ that power can be bought at \$6.50 per horsepower-year and that the cost of producing it is between \$4 and \$5 per horsepower-year.
3. Transportation by water is convenient and cheap.
4. The labor supply is rather poor.
5. Raw materials are plentiful and can be had at a reasonable price.

¹ Joseph W. Richards: The Electrochemical Industries of Norway, *Metallurgical and Chemical Engineering*, vol. ix, No. 10, p. 544 (Oct., 1911).

New York Harbor

The electro-chemical industry of New York harbor is exclusively electrolytic refining of copper bullion.

1. The market for the product is right at hand.
2. Electric power is comparatively costly, being generated locally by steam power.
3. Transportation facilities are of the very best.
4. Labor is plentiful.
5. Raw materials are abundant. This is due to the fact that practically all Western copper smelters now ship their bullion East for refining and that 75 per cent. of the copper of the world is refined near New York.

IV. THE POSSIBILITY OF DEVELOPING ELECTRO-METALLURGICAL INDUSTRIES IN THE WESTERN PART OF THE UNITED STATES

For the purpose of this paper we will consider the manufacturing of the following electro-chemical and electro-metallurgical products:

Aluminum

The process by which alumina is produced consists essentially of refining hydrated alumina, bauxite, to make a pure alumina by a chemical process known as the Bayer process; and then feeding this alumina into a bath of cryolite, forming the electrolyte, with the result that when direct current is passed through the molten mass, molten aluminum is deposited at the cathode and oxygen at the anode.

Raw Materials and Labor.—The ores and other raw materials necessary to the manufacture of aluminum are bauxite, cryolite, coal, and caustic. Such being the case, all the raw materials necessary, with the exception of coal, would, at the present time at least, have to be brought in from some other point, if an attempt were made to manufacture aluminum in any of the intermountain and Pacific Coast States; and in California, it would be necessary to import coal as well. Bauxite is mined in the United States in Alabama, Arkansas, Georgia, and Tennessee. Arkansas is the leading producer. An aluminum plant located at any point in the western part of the United States would be obliged to depend for its source of bauxite upon some of these Eastern States, as bauxite is not known to exist in the Western United States in large quantities. It has been proposed by some that alunite might become a source of aluminum, but all processes for the extraction of aluminum from silicates are still very much in the experimental stage. About 10 per cent. of the bauxite consumed in the United States is imported from Europe. The Southern Aluminium Co., with its plant in North Carolina, proposed to import all of its bauxite from France during the first few years of operation.

The average price of bauxite at the mines, in 1913, was \$4.75 per long ton. A typical bauxite analysis follows:

	Per Cent.
Insoluble.....	12.13
Loss on ignition.....	28.97
Alumina (Al_2O_3).....	57.56
Iron oxide (Fe_2O_3).....	1.34

About 4 long tons of bauxite are necessary per ton of aluminum, and will cost at the mine in Arkansas or Georgia, \$19. This bauxite, if used in the western part of the United States, must be hauled from the East by rail, or by rail and water. It is doubtful if raw bauxite could be laid down at any point in the West for less than \$14 per ton. Therefore the total cost of bauxite per ton of aluminum produced from it would be \$56. Four tons of bauxite would produce 2 tons of alumina. The cost of purification would bring the cost of alumina to \$38 per ton of 2,000 lb. Two tons of alumina produce 1 ton of aluminum, so that the cost per ton of aluminum with bauxite at \$14 would be \$76 per ton.

Freight could be saved by purification of bauxite to alumina at the mine or nearby. The cost of alumina at the mine would be about \$19.50 per short ton, or about \$28 to \$30 in the West. The alumina for 1 ton of aluminum would thus cost from \$55 to \$60, a saving of about \$9 per ton of alumina, or \$18 per ton of aluminum made from the alumina.

In the manufacture of aluminum, 0.1 ton of cryolite (sodium aluminum fluoride (AlF_3 , 3NaF), is consumed by volatilization of the electrolyte per ton of aluminum made. Cryolite is found only in Iceland. Some aluminum manufacturers substitute an artificial fluoride for the cryolite, using also some calcium fluoride, fluorspar. Cryolite is worth about \$24 per long ton, or 1.1c. per pound, in New York. Shipped to the western part of the United States, cryolite would cost about \$35 per long ton, or 1.5c. per pound.

In the electrolysis of alumina in the electric furnace, the oxygen given off at the anode attacks the carbon electrode with the result that about 0.7 lb. of carbon anode is consumed per pound of aluminum produced. In France and Germany these electrodes are manufactured at the larger aluminum plants from coke at the cost of 2.5c. per pound. The cost of manufacturing them in the West with coke costing, at a minimum, \$10 per ton, or charcoal at \$8 per ton, would be about 5c. per pound. Electrodes could be brought from the East or Europe at a cost of 5c. to 6c. per pound f.o.b. San Francisco.

The cost of other materials, such as caustic soda and fluorspar, will not be discussed here, as the amount used is small, and their cost has been included in other estimates.

Labor costs in the West for manufacture of aluminum will be higher than in the Eastern States or Europe. The average wage for laborers at aluminum plants is \$1 per day in Europe, and \$1.50 in the Eastern United States and \$2.50 in the West, outside of mining camps.

Power.—The manufacture of aluminum requires direct current, so

that the use of high-tension alternating currents, as delivered by Western power companies, would mean the transformation of this high current down to a lower voltage for use in a motor-generator set. Allowing for transformer, motor, generator, and line losses, this would result in the delivery to the furnace from the direct-current generator of about 85 per cent. of the energy delivered to the transformers. If we assume that the plant would be operated on a flat rate of only \$10 per horsepower-year for 11,000-volt alternating current, the cost of power at the electrolytic furnace, on the basis of 100 per cent. load factor, would be \$11.76. While it is sometimes asserted that an industry like the manufacture of aluminum can be operated on a 100 per cent. load factor, this cannot be used as a safe figure in calculating costs. The manufacture of aluminum, however, will maintain a higher load factor than some other similar industries, as for instance electric-furnace manufacture of iron and steel, because of the small units employed, and the ease of replacing a broken-down unit with a new one without much additional capital outlay caused by keeping reserve units. The largest aluminum furnaces do not require over 150 hp. for operation, while electric iron and steel furnaces have a capacity of from 300 to 12,000 hp. Another point, influencing the load factor maintained, is the market for the product manufactured. This is especially true for a new concern which has to build up its market. When buying power on a flat rate instead of by the kilowatt-hour, with a variation of load factor say between 85 and 100 per cent., any curtailment of production due to market conditions will result in a heavy charge to the electro-metallurgical company for power which it is not using. For this reason, a load factor of 90 per cent. is assumed in calculating the cost of power. This would increase the cost of direct current at the aluminum urnace to \$13.06 per horsepower-year, or 0.2c. per kilowatt hour.

Cost of Production.—The following estimate of cost of production of aluminum is based upon a plant requiring 25,000 hp. or more, in units of 100 to 150 hp.

Cost of Production of Aluminum per Ton (2,000 lb.) with Conditions as above Stated

2 tons of alumina at \$28.75 per ton.....	\$57.50
200 lb. of cryolite at 1.5c. per pound.....	3.00
1,400 lb. electrodes at 5c. per pound.....	70.00
Other fluxes, etc.....	10.00
28,000 kw-hr. at 0.2c. per kilowatt hour.....	56.00
Labor.....	70.00
Repairs.....	10.00
Amortization, depreciation, 5 per cent. each.....	18.00
Interest, 6 per cent.....	10.00
General.....	20.00
Total.....	\$324.50

per ton, or 16.22c. per pound.

Allowing \$15 per ton for freight to New York and \$20 per ton for marketing expense, the total cost per ton f.o.b. New York would be \$359.50 per ton, or 17.98c. per pound, or in round numbers, 18c. per pound.

Market.—The average prices for No. 1 ingot metal at New York during 1913, generally for spot metal rather than future deliveries, are stated below:

Months	Cents per Pound	Months	Cents per Pound
January.....	26 $\frac{1}{8}$ to 26 $\frac{3}{8}$	July.....	23 $\frac{1}{8}$ to 23 $\frac{3}{8}$
February.....	25 $\frac{3}{4}$ to 26 $\frac{1}{2}$	August.....	22 $\frac{7}{10}$ to 23 $\frac{1}{10}$
March.....	26 $\frac{5}{10}$ to 27 $\frac{1}{10}$	September.....	22 to 22 $\frac{3}{8}$
April.....	27 to 27 $\frac{3}{8}$	October.....	20 to 20 $\frac{5}{10}$
May.....	26 $\frac{3}{10}$ to 26 $\frac{3}{4}$	November (first half)....	19 $\frac{1}{2}$
June.....	24 $\frac{3}{4}$ to 25 $\frac{1}{4}$	November (last half)....	19 $\frac{1}{8}$ to 19 $\frac{3}{8}$
		December.....	18 $\frac{7}{10}$ to 19 $\frac{1}{10}$

The tariff act enacted in October, 1913, placing a duty of 2c. per pound on ingot aluminum instead of the previous duty of 7c. per pound, resulted in the price falling in October of that year to from 20c. to 20.5c. The above quotations are indicative of transactions in the open market, especially as made by dealers in foreign aluminum. The average price of contract aluminum was 1c. to 2c. per pound less than the prices given. Hence it is evident that in order to avoid a shutdown in dull times, a company manufacturing aluminum in the western part of the United States should be able, if necessary, to sell its aluminum in the New York market at a price of 18c. per pound.

Aluminum manufactured in the West would at present have to compete with a domestic production of 45,000,000 lb. of aluminum, and imports of about 30,000,000 lb. It is asserted that the domestic production of aluminum will in the future be increased by at least 20,000,000 lb. per annum, due to the Southern Aluminium Co. starting its plant, so that inside of two years all of the present imports except 10,000,000 lb. will be supplied by domestic producers. It is asserted also that the Aluminum Co. of America contemplates doubling its present capacity at a plant in Tennessee. If this be done, it will eventually add a possible production of 40,000,000 lb. per annum, or a total future output of 30,000,000 lb. more than the present market demands. Thus it is clear that any company starting business now will have strong competition to meet several years hence, or, in other words (since its plant could not be completed for several years), from the very start of its operation. This competition would exist in spite of the fact that the consumption of aluminum in the United States has increased at the average rate of about 7,000,000 lb. per annum for the last five years. While at this rate of increase of consumption the contemplated increased output would no more than supply the domestic demand, it is not probable that the con-

sumption will maintain quite as high a rate of increase. Moreover, by reason of the decreased tariff, foreign manufacturers will also be much more active in the New York market hereafter.

Conclusions.—In view of the above statements, we are of the opinion that the manufacture of aluminum in the western part of the United States would not prove profitable.

Even if electric power could be had at \$10 per horsepower-year, the high freight charges on raw materials and product and increased labor costs over Eastern wages are such as to raise the cost of production so high that in periods of depressed market it would about equal the lowest selling price. In this connection it should be remembered that by reason of decreased output manufacturing costs invariably increase in times of business depression. Since such a plant would buy power on a flat-rate basis, any decrease in output would result in a greater increase of unit cost, due to the continuing charge for power regardless of the amount used.

A study of the preceding data shows that freight rates on raw materials, supplies, and product would contribute largely to the cost of the production of aluminum in the West. While India bauxite might be shipped to the West very cheaply, little is known about these deposits, and their development is in the distant future. An example of the cost of materials due to freight is in electrodes. They can be purchased in the East at 3c. per pound. On the basis of 0.7 lb. consumption of electrode per pound of aluminum, the cost of electrodes at 5c. per pound increases the cost of aluminum by 1.4c. per pound.

Besides the higher labor costs due to increased wages in the West, technical men, such as chemists and engineers, also receive a higher rate of pay.

At the low price of aluminum in 1913, a company manufacturing aluminum in the West could about "break even" if it sold its product. At a price of 20c. per pound a profit of 2c. per pound could be made. A production of 10,000,000 lb. per annum would require a plant costing about \$4,000,000. This production would give a profit of \$200,000 per annum, or 5 per cent. on the actual investment. This is generally considered much too low a margin of profit for safe investment, since unforeseen conditions might arise to increase the cost. Estimates previous to operation are usually found in practice to be too low. It is probable that the Eastern producer could sell at a profit at 18c. per pound; since it has been estimated that the cost of production of aluminum at Niagara Falls is about 15c. per pound.

A company now producing aluminum, with a market for its product and an intimate knowledge of the industry, could possibly establish a manufacture in the West; but we see no reason why such a company should go to the West for such a purpose at the present time, or in the very near future, at least so long as it is able to secure power near the ore and the market,

as at the present time. Apparently, an established company owning its bauxite supply and having a market, is the only sort of a concern which would be able to operate a plant in the West without losing money, and it could not make as much there as in an Eastern plant.

Pig Iron

In the electric-furnace smelting of iron ores, electric energy, instead of charcoal or coke, as in the blast furnace, is used to supply the heat necessary for smelting; and as only one-third or less of the charcoal or coke that is used in the blast furnace is needed in the electric furnace for the reduction of iron oxides, it is possible to produce in the electric furnace three times as much iron with one ton of charcoal or coke as in the blast furnace.

Raw Materials and Labor.—The raw materials necessary for the manufacture of pig iron in the electric furnace are iron ore (either hematite or magnetite), limestone, and charcoal or coke. There is no choice as to the kind of iron ore, as the electric furnace will handle either hematite or magnetite. In regard to the limestone, it is sometimes considered advisable to calcine the limestone before use in the electric furnace, in order to save the energy necessary to remove the carbon dioxide from the limestone in calcination; but the use of calcined limestone is not advisable because of the fine material added to the charge in this way. Charcoal is preferable to coke in the electric-furnace manufacture of pig iron, because the energy consumption is smaller (because of the possibility of using the shaft type of furnace with charcoal), and operation more steady, Coke can be used, however, in the rectangular type of furnace without a high shaft.

Iron ore of good grade is more or less plentiful throughout the Western States, but unfortunately the cost of transporting it to a point where it could be utilized might prohibit its use. Moreover, it is estimated that Chinese iron ore containing 60 per cent. of iron can be now laid down at Pacific Coast ports at a cost of about \$5 per ton. Such being the case, we will assume that the cost of iron ore would be \$5.

Limestone deposits are also more or less abundant throughout the West, and so the cost of lime as a flux in the production of pig iron would probably not be prohibitive.

As already observed, charcoal or coke could be used as a reducing material. At the starting of a plant, charcoal would be the preferable material, but if the plant assumed great proportions, say requiring much over 100 tons of charcoal per day, we believe that it would be necessary to use coke, on account of possible scarcity of charcoal. For the purpose of this paper, we will assume the cost of the reducing agent as not less than \$10 per long ton, or \$3.33 for about one-third of a ton, required for reduction per ton of pig iron produced.

Labor requirements are the same as in the blast-furnace manufacture of pig iron, and a minimum wage of \$2.50 per day of 8 hr. is assumed.

Power.—Power will be considered as costing at the furnaces \$11.70 per horsepower-year, or 0.18c. per kilowatt-hour, allowing for transformer losses in reducing the voltage to from 40 to 110 volts.

Cost of Production.—The following estimate is based upon an annual production of 50,000 tons of pig iron. The plant would consist of five electric furnaces of 3,000 hp. each, the whole plant requiring 18,000 hp., including 500 hp. for various uses outside of the furnaces.

Estimated Cost per Long Ton of Producing Pig Iron in the Electric Furnace, with Conditions as above Stated

1.6 tons of iron ore at \$5 per long ton	\$8.00
0.33 long ton of charcoal or coke at \$10	3.33
0.25 long ton of limestone at \$1.75	0.44
10 lb. of carbon electrodes at 5c.....	0.50
2,400 kw-hr. at 0.18c.....	4.32
Labor.....	5.00
Maintenance and repairs.....	0.50
Amortization, depreciation, at 5 per cent. each.....	1.70
Interest at 6 per cent.....	1.02
General.....	1.40
Total.....	\$26.21

Based upon the above estimate, the cost of producing pig iron by use of the electric furnace would be about \$26 per long ton. As the market for this pig iron would be entirely upon the Pacific Coast, it would have to compete with iron brought from the East and with foreign iron.

Market.—The market for this pig iron manufactured in any of the intermountain or Pacific Coast States would largely be a local one. At the present time, this market would probably not be very great, since with pig iron selling on the coast at from \$20 to \$25 per ton, there is little incentive to use it in foundry work, when scrap iron can be purchased at a much lower price. There is no large steel plant on the coast which would be a consumer of pig iron. Of course, with a production cost f.o.b. Pacific Coast point of \$26 per long ton, little if any profit could be made on pig iron produced there, as the price ranges nearer \$21 than \$25 per ton f.o.b. San Francisco.

Another important factor lies in the cheapness with which pig iron from England, China, and India can be laid down upon the Pacific Coast. Pig iron from any of these countries could be delivered at Pacific Coast ports for from \$18 to \$20 per long ton. With the Panama Canal open, it is now possible to lay Eastern pig iron down on the Pacific Coast for about \$18 per long ton, so that the foreign or Eastern producer could considerably undersell the Western manufacturer.

Conclusions.—The electric-furnace manufacture of pig iron in any of the Pacific Coast States does not appear to be feasible commercially because of the high cost of raw material, and market conditions. Nor would it be feasible commercially to produce pig iron for conversion into steel, because Eastern billet steel can be brought to the Pacific Coast at a total cost of \$30.50 per long ton, which would probably be much less than it would cost to produce it in the localities above mentioned.

The high cost of production is largely caused by the high cost of iron ore, which is about double that paid by Eastern blast furnaces, which are producing pig iron for about \$10 per long ton; and also by the high labor cost, which is about 70 per cent. more expensive than Eastern labor. The cost of reducing material is also high as compared with the cost of Eastern coke at the furnaces, in spite of the fact that the electric furnace uses only one-third of the amount of coke used by the blast furnace. It would therefore seem that the cost of production alone would prohibit the success of an electric-furnace pig-iron industry in most of the Western States.

To assure commercial success, the plant should be able to sell its product in the coast market, if necessary, at as low a figure as \$18 per ton. Eastern or foreign pig iron can be laid down at that cost. It is also quite likely that the market for pig iron, with the price as high as \$20 per ton, will not grow very rapidly, as it is cheaper to use scrap iron to make castings.

Steel

There are two forms of steel manufacture in which the electric furnace has been used: (1) cold scrap iron and steel of either inferior or high-grade quality are melted and refined in an electric furnace with the production of steel of the highest grade and equal to the best crucible steel; and (2) molten steel, the product of either the acid or basic converter, or of the acid or basic open-hearth furnace, is super-refined, or made into alloy steel, in an electric furnace. The steels thus made may be cast into ingots, or directly into various shapes. It has been proposed to use the electric furnace for the manufacture of steel from molten pig iron; but with pig iron at \$20 per ton, the cost would be prohibitive. Steel made from molten electric-furnace pig iron would cost at least \$34 and probably over \$40, while that made out of Eastern pig iron, which would have to be melted, would cost at least \$30 and probably \$35, and so could not compete with Eastern steel, which can be sold in the West for about \$30.

As the high cost of pig iron prohibits the establishment of a tonnage steel plant, we will consider only an electric-furnace plant for the production of high-grade steel castings and shapes, and bar steel.

Raw Materials and Labor.—The principal raw material used in the

electric-furnace manufacture of steel is scrap steel. While some scrap cast iron could be used, most of the material melted should be steel or wrought iron. Iron turnings, which in the open hearth are not especially desirable on account of oxidation losses, are about the most adaptable material for use in the electric furnace. There is not the high oxidation loss in the electric furnace that there is in the open hearth. Any scrap material used in the electric furnace must be small in size because of difficulty in operation of a furnace on large scrap iron, due to short circuits. A large part of the turnings produced in the various foundries and shops throughout the West go to waste at present, and could be obtained cheaply.

Power.—In the electric-furnace manufacture of steel, electric energy is used at about the same voltage as in the production of pig iron or ferro-alloys; but the electric steel furnace cannot maintain as high a load factor as the electric iron-smelting furnace or the ferro-alloy furnace. For example, if it be necessary to transform an 11,000-volt current down to 40 or 100 volts, a transformer and line loss of 5 per cent. will occur. This, on the basis of 100 per cent. load factor and with power at \$10 per horsepower-year, would make the power cost \$10.52 at the furnace. Owing to the intermittent nature of the electric-furnace process in steel manufacture, a load factor of over 80 per cent. could be maintained only with difficulty. The power cost would then be \$13.15, or 0.20c. per kilowatt-hour. If a lower rate were made on such power, an electric steel furnace could use peak power to advantage, on account of the intermittent nature of the process.

Cost of Production.—The following estimated cost of production of electric-furnace steel is based upon an annual production of 25,000 tons and the utilization of 4,000 hp. There are so many combinations of furnaces of different sizes with which a plant could be equipped that we will not attempt to specify them. The estimate is based upon steel cast into ingot form.

Cost of Production of Steel in the Electric Furnace in the Western United States

1.1 tons of scrap at \$15 per ton.....	\$16.50
Slag materials.....	1.00
Ferro-alloys	1.00
800 kw-hr. at 0.20c.....	1.60
Labor	2.50
Maintenance and repairs.....	2.40
20 lb. of electrodes at 5c.....	1.00
Amortization and depreciation at 5 per cent. each.....	1.50
Interest at 6 per cent.....	0.90
General	1.00
Royalty	0.50
Total.....	29.90

With conditions as above stated, the cost of production of ingot steel in the electric furnace would be about \$30 per long ton, to which would have to be added the freight rates to the point of market or delivery.

Market.—There is very little market for billet and ingot steel on the Pacific Coast, and at the present time a tonnage steel plant manufacturing ingots for rails and heavy steel would probably not be able to compete in the market with Eastern or foreign products. But there is a considerable demand for small shapes; and a plant in the West casting steel into such shapes might possibly be able to dispose of its product; but a large part of the steel shapes used in the West belong to machinery manufactured in the East, and most of this steel comes from the Eastern States.

Conclusions.—Considering the conditions above stated, it does not look as if the manufacture of tonnage electric-furnace steel in the West would be profitable. As in the case of pig iron, steel in the form of ingots, billets, rails, or structural shapes can be laid down so cheaply on the Pacific Coast and from the Eastern States or foreign countries as to render impossible the profitable manufacture of this steel on the Pacific Coast at the present time. While a small electric-furnace foundry, manufacturing special steel shapes and using 500 to 1,000 hp., might be successful, it would not amount to much as a large-scale steel producer. Tonnage steel cannot be made at a profit on the Pacific Coast at the present time, by reason of the high cost of producing pig iron; and for this reason it would be impossible to successfully operate a self-contained iron and steel plant, using either electric or combustion furnaces. As has been pointed out already, the high cost of producing pig iron is largely due to the high cost and scarcity of a satisfactory reducing agent.

Copper

As already observed, the electric smelting of copper ores is entirely in the experimental stage. Before it can be practiced commercially, much money and time must be spent on experimental work. While we believe it to be feasible under favorable conditions, the process has not yet reached a stage of development that would warrant us in regarding it as fully worked out. When some of the Alaskan deposits have been more developed, electric copper smelting may prove profitable at points along the Alaskan coast, since the ore in this case could be hauled largely by water. As the process itself is not fully developed, we do not consider it worth while to give an estimate of the cost of smelting copper ores by electricity.

Zinc

While the electric smelting of zinc ores is more advanced than that of copper ores, it has not been yet proved a commercial, or, in fact, a tech-

nical success. Such being the case it cannot, at present, be seriously considered as a consumer of power. However, the process of electric zinc smelting will doubtless be more fully developed in the near future, and it is quite likely that the electric smelting of zinc ores will consume considerable power, but at present, as in the case of copper, we do not consider the process sufficiently developed to warrant estimates of cost.

V. GENERAL CONCLUSIONS

1. The market for hydro-electric power for electro-metallurgical industries in general is not great; and under the best of conditions the part consumed in electro-metallurgical plants would be a very small proportion of the total hydro-electric power that it is possible to develop in the Western States.

2. Although the outlook at the present time is not favorable to the establishment of extensive electro-metallurgical industries, we are nevertheless of the opinion that such industries will, in time, be established, but perhaps along other lines than at present, that is, other than the production of aluminum, ferro-alloys, pig iron, steel, etc.' It is also doubtless true that the electric furnace will in time be quite generally used for the local production of steel castings, but an extensive use of the electric furnace for this purpose will not involve any great consumption of electric power.

Therefore, if our analysis of the situation be correct, is it reasonable to expect that the hydro-electric power companies may ultimately be able to dispose of a great portion of their surplus power by reason of the development of new electro-metallurgical industries? We believe they will, and for this reason:

At the present time the metallurgy of the non-ferrous metals is rapidly changing. The processes which were suited to the treatment of non-ferrous ores five or ten years ago are at the present time not satisfactory, because the non-ferrous metallurgical plants of the country are called upon to treat ores of a lower grade, and also more complex than formerly. Such being the case, processes must be devised which will meet these requirements. However, we do not for a moment imagine that electro-thermic or electrolytic processes will prove to be the only solution of the many problems which are at the present time confronting the non-ferrous metallurgist. Although such processes may greatly assist in solving these problems, we are of the opinion that further research work will indicate other solutions. This is true, especially as regards hydro-metallurgical processes, because the profitable treatment of the low-grade and complex ores above mentioned requires the use of cheap reagents. Such being the case, the hydro-metallurgical treatment of such ores may bring about the establishment of an electro-chemical industry for the produc-

tion of the necessary reagents and thus indirectly the metallurgical industry may bring about an extended use of hydro-electric power. Whether this will prove to be the case or not, remains to be seen. It can only be determined by a careful investigation of the subject, and by extensive research, having for its object the finding of new uses for electricity in metallurgical work.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Tonopah Plant of the Belmont Milling Co.

BY A. H. JONES,* TONOPAH, NEV.

(San Francisco Meeting, September, 1915)

THE Belmont mill at Tonopah, Nev., was designed and constructed by the Belmont staff. Ground was broken in August, 1911, and milling operation started July 25, 1912. The metallurgical flow sheet, and the machinery adapted to carrying out operations in accordance with this plan, were decided upon from experience gained in the treatment of Belmont mine ores at the old Belmont mill at Millers, Nev.

The detail drawings were made by the engineering staff of the Tonopah-Belmont Development Co., under the supervision of Otto Wartenweiler, who also had direct charge of the construction, consulting with the General Superintendent of the company, Frederick Bradshaw, and the writer.

TABLE I.—Construction Costs, Belmont Mill

	Excavation Concrete Walls and Founda- tions	Floors and Machinery Founda- tions	Buildings		Machinery, Including Erection, Piping, Wiring, Belting, Etc.	Totals
			Frames	Covering		
Crusher plant.....	\$5,760.72	\$2,527.58	\$2,230.72	\$1,476.69	\$21,174.96	\$33,170.67
Inclined conveyor.....	166.41	238.02	1,771.75	585.19	3,620.79	6,382.16
Battery bins.....	399.00	489.70		2,067.69	3,475.14	6,431.83
Stamps.....		8,797.82			36,873.06	45,670.88
Tube mills and classifiers.....		3,180.95			40,127.89	43,308.84
Callow cones.....		26.02			856.63	882.65
Concentrating plant.....		1,663.23			11,356.35	13,019.58
Concentrate house.....	76.80	449.69	354.80	623.27	602.10	2,106.66
Dorr thickeners.....		11,297.98			15,034.83	26,332.81
Circulating system.....		147.81			6,846.05	6,993.86
Air agitation.....	261.71	3,987.80			25,257.89	29,507.40
Clarifying.....		1,829.63			8,573.87	10,403.50
Precipitation system.....	395.00	44.05			29,581.02	30,020.07
Briquetting plant.....					1,589.49	1,589.49
Air compressor.....		1,084.39			7,094.62	8,178.91
Filter plant.....		4,456.68			30,104.60	34,561.18
Refinery.....	2,473.84	1,552.23	2,200.17	2,292.80	7,548.79	16,067.83
Boiler plant and fuel-oil system.....	571.56	90.97	401.25	581.11	7,606.28	9,201.17
Tank-heating system.....		9.56			3,362.39	3,371.95
Transformer house.....	91.83	101.05	428.69	213.07	4,804.89	5,639.53
Lime house.....	11.00			753.30	305.85	1,070.15
Machine shop.....	1,297.66	843.01	1,138.73	1,191.93	339.36	4,810.69
Storeroom.....	511.55	1,509.60	1,305.84	1,315.78	152.04	4,794.81
Inclined railway.....	133.25		432.70	203.41	379.51	1,148.87
Mill building.....	39,645.45	6,757.60	45,493.48	19,607.14	9,020.51	120,524.18
Total.....	\$51,795.78	\$51,085.37	\$55,758.13	\$30,861.38	\$275,688.41	\$465,189.07

* Superintendent of Mills, Tonopah-Belmont Development Co.

The total cost of the plant, designed to handle 500 tons per day, was \$465,189.07. The saving in operating costs and increased extraction, as compared with the best results from the old mill at Millers, was such as to return the entire cost of the new plant from the treatment of 260,000 tons of ore, or approximately 18 months' operation. Fig. 1 is a view of the plant, and Fig. 2 a section of the mill.

Segregated construction costs are given in Table I.

In order to present the matter clearly, a general description of each step of the milling operation will first be given and the metallurgical process considered, after which the operation will be gone over again, step by step, and discussed from a mechanical and economic standpoint. The general flow sheet is given in Fig. 3.

GENERAL DESCRIPTION AND METALLURGY

Crushing and Conveying

All crude ore is broken underground to pass a 9-in. grizzly and hoisted in 3-ton skips to the ore pockets, 37 ft. above the collar of the shaft. From the ore pockets it is conveyed in a 2-ton car by a Hunt automatic railroad to the crusher pockets.

Two circular steel crusher pockets, with a combined capacity of 1,000 tons—about 500 tons of which will run freely—are situated one on each side of the picking belt. Rock is fed from the crusher pockets through finger gates, hand operated, over shaking grizzlies with 2-in. openings. The undersize bypasses the picking belt, going direct to the trommel, and the oversize is fed to a steel picking belt 40 in. wide, with 50-ft. centers, traveling 45 ft. per minute, from which belt seven ore sorters pick the waste and throw it down chutes on to a 20-in. conveyor belt which discharges upon the waste dump.

The picking belt discharges over the shaking feeder to a No. 7½K gyratory crusher set at 2-in. ring. The discharge from this crusher joins the undersize from the crusher pockets to form the total feed for the trommel, which is 48 in. in diameter by 14 ft. long, with 1¼-in. openings.

The undersize from the trommel goes direct to the inclined conveyor belt, and the oversize forms the feed for two No. 4D short-head gyratory crushers set to 1-in. ring.

The discharge from the No. 4 crushers and the undersize from the trommel are delivered to a 20-in. conveyor belt, with 250-ft. centers, on an incline of 19° 20', which carries the ore to the head of the mill.

While the ore is passing over this belt it is automatically weighed by an electric weighing machine.

From the head of the incline the ore is conveyed and distributed, by means of a horizontal belt conveyor and automatic tripper, to flat-bottom

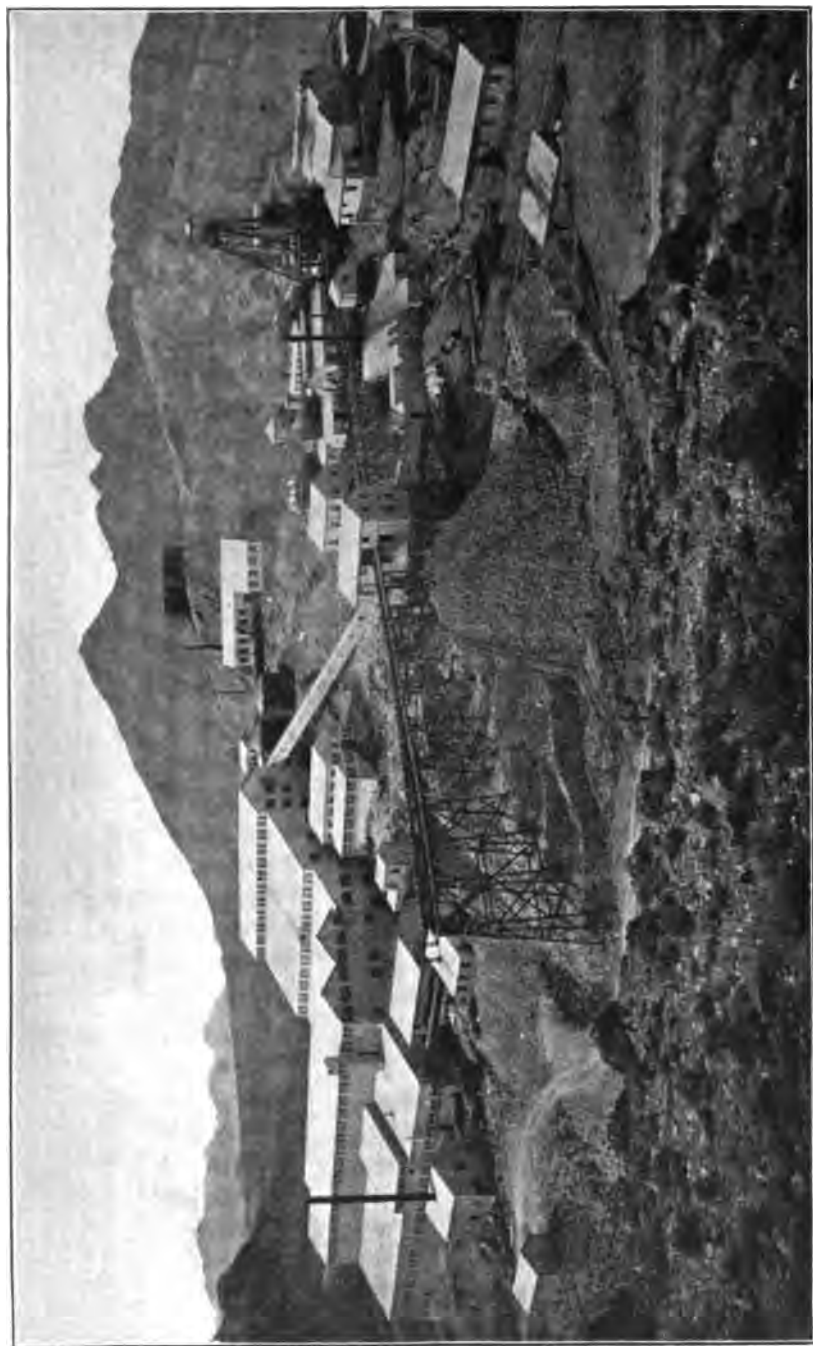


FIG. 1.—VIEW OF THE PLANT OF THE BELMONT MILLING CO.

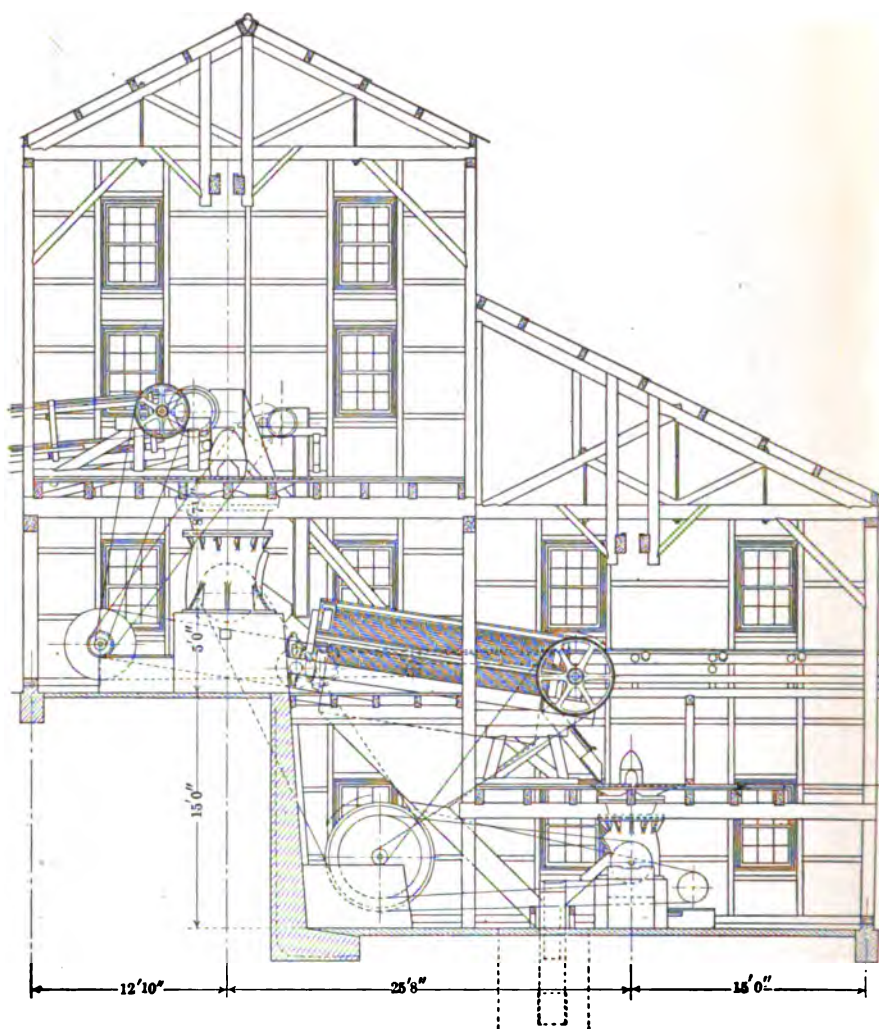


FIG. 2.—SECTION OF BELMONT MILL.

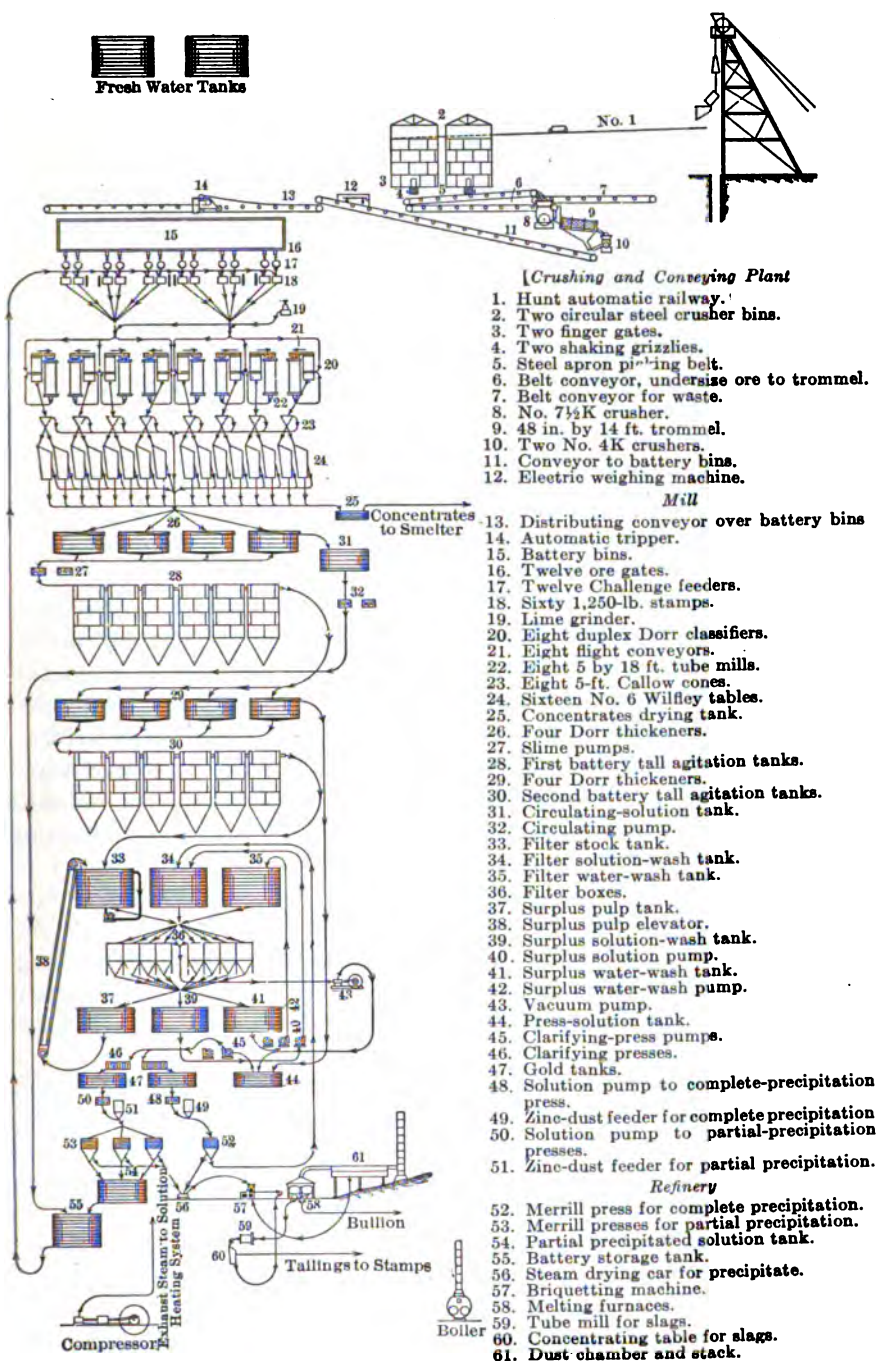


FIG. 3.—FLOW SHEET OF BELMONT MILL.

battery bins, 16 by 17 by 110 ft. in the clear, with a capacity of 1,500 tons, about one-half of which will run freely.

Sampling

At the point where the inclined belt discharges on to the horizontal belt is placed a device consisting of a 74-tooth gear with 24 teeth removed and a 25-tooth pinion, with a bucket of about $5\frac{1}{2}$ lb. capacity attached to the gear. The gear makes one revolution every 68 sec., traveling very slowly until the bucket reaches the stream of ore, when a counterbalance actuates the gear through the part with the teeth removed, allowing the bucket to make a quick cut of the ore. The sample thus taken is further crushed in a McCully laboratory gyratory crusher to pass $\frac{1}{4}$ -in. ring and riffled down to about 50 lb., which is further reduced and cut at the assay office for a representative sample of the ore crushed during the day. While this original cut, of about 1 ton in 500, seems small, it was checked for 3,500 tons against the sampling of the Western Ore Purchasing Co. and averaged within 8c. per ton of their results.

As an accurate sampling with this grade of ore cannot be had without a very elaborate sampling plant, and at a considerable operating cost per ton, while the tailings or discharge from the mill is a product easily sampled, being thoroughly mixed and of such low value that a large error is not possible, the actual value of the ore treated for any month is figured from bullion produced, plus concentrates made, plus tailings discharged. The daily sampling of ore milled is entered in the mill books and used simply as a check against the other method of figuring the metallurgical results.

Stamping

From the battery bins the ore is fed through open gates and Challenge feeders to sixty 1,250-lb. stamps, which crush through 4- and 6-mesh Tyler "ton-cap" screens. For the fiscal year Mar. 1, 1914, to Feb. 28, 1915, the average stamp duty was 8.88 tons per day.

Stamping is done in cyanide solution, about 5 parts of solution to 1 of ore being introduced into the mortar, the solution having a cyanide strength of 5 lb. and a lime strength of about 1 lb. CaO. The discharge is kept about 3 in. above the dies. A screen analysis, with one-half 4-mesh and one-half 6-mesh screens, is given on the following page.

The launders for the 30 stamps on each side are so arranged, the discharge lip of each battery of five stamps being divided at the center, that any $2\frac{1}{2}$ stamps can deliver to any one of the four tube mills on that side of the mill. This is accomplished without an elaborate or cumbersome launder system by carrying a four-compartment launder along the retaining wall below the stamps, each compartment delivering to its respective

tube mill. From each $2\frac{1}{2}$ stamps there is a short, closed-end launder, with a hose discharge that can be changed to any compartment of the main launder system.

Screen Analysis of Battery Discharge

Mesh	Per Cent.
+ 10.....	10.0
+ 20.....	19.0
+ 40.....	22.0
+ 80.....	15.0
+100.....	3.5
+150.....	4.0
+200.....	2.5
-200.....	24.0
	100.0

Tube Milling

The battery discharge is delivered to eight Dorr duplex classifiers, placed parallel with their respective tube mills, and at such an elevation that a spiral feeder with a 36-in. lift will deliver coarse feed to the mill and at the same time allow the tube discharge, together with the battery discharge, to flow, with no other elevation, into the classifier and form a closed circuit.

Sufficient lime for one 8-hr. shift is slaked in a grinding pan, used as a mixer, and the overflowing milk of lime is introduced with the tube-mill feed. At this point about 0.15 lb. of lead acetate—approximately one-half the total amount used in the mill—is added.

For some time after the mill commenced operation, the grinding was kept well above 80 per cent. through 200 mesh, with eight mills, but experimental work has since demonstrated that the higher extraction obtained with 85 per cent. through 200 mesh, over 75 per cent. through 200 mesh, would not compensate for the cost of the further reduction, and we are at this time finishing about 75 per cent. through 200 mesh, with seven mills.

There has been a great deal of controversy as to the amount of return feed when running tube mills in closed circuit, and to check this return a central baffle was put into a Dorr duplex classifier, giving the initial feed to one side and the return to the other side, and the tonnage from both sides carefully calculated. This test gave the following figures: Total feed 127.38 tons, initial feed 67.18 tons, return feed 60.20 tons. Ratio of return to initial feed 0.91 to 1. Screen analyses and tonnages are given in Table II.

TABLE II.—Tonnes and Screen Analyses for Classifier and Tube Mill in Closed Circuit

Mesh	Battery Discharge Tonnage by Weighed Sample			Battery Tube Feed Tons by Weighed Sample			Battery Dorr Overflow Tonnage by Difference between Battery Discharge and Battery Tube Feed			Tube-Mill Discharge Tonnage by Battery Feed and Return Feed			Return Feed Weighed Feed			Return Dorr Overflow Tonnage Battery Feed to Tube			Dorr Overflow General Battery Discharge Tonnage		
	Cumulative, Per Cent.	Tons	Cent.	Cumulative, Per Cent.	Tons	Cent.	Cumulative, Per Cent.	Tons	Cent.	Cumulative, Per Cent.	Tons	Cent.	Cumulative, Per Cent.	Tons	Cent.	Cumulative, Per Cent.	Tons	Cent.	Cumulative, Per Cent.	Tons	Cent.
+ 10	8.2	6.02	8.2	8.2	5.51	8.2	8.2	5.51	8.2	8.2	5.51	8.2	8.2	5.51	8.2	8.2	5.51	8.2	8.2	5.51	8.2
+ 14	12.3	20.5	9.03	13.1	21.3	8.80	13.1	21.3	8.80	13.1	21.3	8.80	13.1	21.3	8.80	13.1	21.3	8.80	13.1	21.3	8.80
+ 20	13.8	32.3	8.68	12.2	33.3	8.19	12.2	33.3	8.19	12.2	33.3	8.19	12.2	33.3	8.19	12.2	33.3	8.19	12.2	33.3	8.19
+ 28	15.2	44.5	8.95	13.1	46.0	8.50	13.1	46.0	8.50	13.1	46.0	8.50	13.1	46.0	8.50	13.1	46.0	8.50	13.1	46.0	8.50
+ 35	18.0	53.4	6.53	9.8	56.1	6.58	9.8	56.1	6.58	9.8	56.1	6.58	9.8	56.1	6.58	9.8	56.1	6.58	9.8	56.1	6.58
+ 48	27.9	61.3	3.79	9.1	65.1	6.12	9.1	65.1	6.12	9.1	65.1	6.12	9.1	65.1	6.12	9.1	65.1	6.12	9.1	65.1	6.12
+ 65	5.1	66.4	3.74	7.0	73.5	4.70	Trace	73.5	4.70	Trace	73.5	4.70	Trace	73.5	4.70	Trace	73.5	4.70	Trace	73.5	4.70
+ 100	5.0	71.4	3.67	10.0	82.5	6.72	2.5	82.5	6.72	2.5	82.5	6.72	2.5	82.5	6.72	2.5	82.5	6.72	2.5	82.5	6.72
+ 150	4.4	75.8	3.23	10.5	93.0	7.05	9.125	93.0	7.05	9.125	93.0	7.05	9.125	93.0	7.05	9.125	93.0	7.05	9.125	93.0	7.05
+ 200	2.2	78.0	1.61	2.2	95.2	1.48	6.0	95.2	1.48	6.0	95.2	1.48	6.0	95.2	1.48	6.0	95.2	1.48	6.0	95.2	1.48
- 200	22.0	100.0	16.15	4.8	100.0	3.23	82.375	100.000	5.108	31.625	100.000	40.29	8.17	100.00	4.92	74.25	100.00	49.9	79.5	100.0	58.35
Total Tons		73.37			67.18			6.200			127.38			60.21			67.20			73.40	

A simple and accurate method of figuring this return tonnage, in accordance with the screen table below, is as follows:

Mesh	Battery Discharge		Tube-Mill Feed		Tube-Mill Discharge		Dorr Overflow	
	Per Cent.	Cumulative, Per Cent.	Per Cent.	Cumulative, Per Cent.	Per Cent.	Cumulative, Per Cent.	Per Cent.	Cumulative, Per Cent.
+ 10	10.3	10.3	5.4	5.4				
+ 14	10.0	20.3	5.2	10.6				
+ 20	10.2	30.5	5.3	15.9				
+ 28	11.3	41.8	5.9	21.8				
+ 35	7.7	49.5	4.9	26.7	1.0	1.0		
+ 48	7.1	56.6	6.5	33.2	3.1	4.1		
+ 65	5.6	62.2	8.1	41.3	6.5	10.6		
+100	6.0	68.2	20.0	61.3	20.0	30.6	1.0	1.0
+150	6.6	74.8	24.4	85.7	28.6	59.2	12.5	13.5
+200	4.0	78.8	7.8	93.5	12.7	71.9	14.2	27.7
-200	21.2	100.0	6.5	100.0	28.1	100.0	72.3	100.0

Tons crushed per day, 500. Feed per mill, 62.5 tons.

In the battery discharge there was 41.8 per cent., or 26.12 tons, which did not show in the tube-mill discharge. This material was represented in total tube-mill feed by 21.8 per cent. As we know this 21.8 per cent. is represented by 26.12 tons, 100 per cent. of the feed amounts to 119.8 tons.

Total feed, tons.....	119.8
Initial feed, tons.....	62.5
Return feed, tons.....	57.3
Ratio of return to initial feed.....	0.92 to 1

Concentration

The overflow from the Dorr classifiers, or the finished product from the tube mills, flows to eight 5-ft. Callow cones, used as sloughing-off cones, the overflow going from there direct to the Dorr thickeners, and the spigot discharge forming the feed for 16 No. 6 Wilfley concentrating tables, running 233 strokes per minute with a $\frac{5}{8}$ -in. throw.

Close concentration is not attempted, the intention being to take out only the heaviest and most refractory material. This method gives a very clean concentrate product, but not a close concentration.

The concentration for the fiscal year Mar. 1, 1914, to Feb. 28, 1915, was 0.67 per cent. by weight and 11.5 per cent. in extraction of precious metals.

The concentrates are trammed from boot boxes to a vacuum tank, 14 ft. in diameter by 3 ft. deep, so arranged that it forms the bed of a 15-

ton card-recording scale, and built over a concentrate storage bin. In this tank the concentrates are allowed to dry by vacuum for 48 hr. when they are weighed and shoveled through a center gate to the steel bin below and sampled for both moisture and metallic content. The elevation of the floor of this bin is the same as that of a railroad freight car. At the time of shipment the entire lot is sampled by saving every fifth shovel, this sample being cut by saving each third shovel, leaving about 3 tons, which is quartered down to final sample, the reject at all times being wheeled direct to the freight car. By this method sampling and loading are finished at the same time.

A great deal of experimental work has been done with a view to treating concentrates on the ground, and while a very attractive extraction can be obtained with new solution, without regeneration the solution would soon become so foul and inactive that a satisfactory extraction could not be made. And even under the most favorable conditions the cost of treatment would be slightly higher than the shipping and marketing expense. It should be stated in this connection that after roasting concentrates a satisfactory extraction cannot be obtained with cyanide.

An analysis of concentrates is given below:

	Per Cent.
S.....	31.6
Insoluble.....	30.6
Fe.....	29.8
Pb.....	1.3
Zn.....	0.6
CaO.....	0.8
Mn.....	1.1
Cu.....	0.6
Au and Ag.....	1.6
Al ₂ O ₃	0.42
Undetermined.....	1.58
Total.....	100.00

First Thickening

The pulp from the Wilfley tables and the overflow from the Callow cones flow to four 30 by 12 ft. Dorr thickeners. The ratio of solution to ore is about $8\frac{1}{2}$ to 1, added as follows: 5 or 6 parts at the batteries, about 0.8 for correction of moisture at the tube-mill feed, about 1 part at the Dorr classifiers in order to thin the pulp for proper classification, and about 1 part as wash water on the concentration tables.

The four 30 by 12 ft. Dorr thickeners allowed 5.5 sq. ft. of settling area per ton of ore, and this was adequate for the first year of operation, but since then development in the mine of veins narrower than the main Belmont vein—and from which a large tonnage has been produced—has

tended to send to the mill a material much more talcose than formerly, and to overcome this difficulty Dorr settling trays were installed in two of the four primary thickeners, allowing, on a 500-ton per day basis, 8.47 sq. ft. of settling area per ton of ore every 24 hr.

These settling trays have worked out to very good advantage, giving no serious trouble in a mechanical way and increasing the capacity of the two 30 by 12 ft. thickeners in which they are installed by about 75 per cent.

Agitation

The overflow from the lower or first thickeners flows either to the circulating tank or to the precipitation supply tanks.

The underflow, at about 1.26 specific gravity, is pumped by an 8½ by 10 in. Aldrich pump (with a Campbell & Kelly steel-lined centrifugal pump as a reserve) to the No. 1 air agitator. The flow is continuous through the first battery of six agitators—each 15 ft. in diameter by 45 ft. deep, with 55° cone, and having a capacity of 6,000 cu. ft. Agitation is effected by a central air lift. From the No. 6 agitator the pulp is elevated by an air lift to a height of 6 ft. above the tank and delivered to a system of launders, where it is further diluted with about 4 parts of partly precipitated solutions, and thence flows to four 30 by 12 ft. Dorr thickeners to be again thickened, the overflow going by gravity to the precipitation supply tanks. The thickened pulp is delivered by air lifts to the No. 7 agitator and flows continuously through the second battery of six agitators to the 28 by 20 ft. filter stock tank, equipped with a Trent agitator to keep the pulp from settling. The total period of agitation, under normal conditions, is about 48 hr.

By thus thinning the pulp with barren solution after it has passed through the No. 6 agitator and re-thickening, a large proportion of the dissolved silver and gold is displaced, which relieves the filters from this work and allows a change of solution for the final agitation.

Enough cyanide is introduced into the No. 1 agitator to bring the strength up to 6 lb. per ton of solution; about 0.18 lb. of lead acetate—a little over half the total quantity used in the mill—is also added.

About 2 per cent. better extraction is obtained with heated solutions than with cold, and, therefore, the solutions in the agitators are heated to a temperature of about 90° F. by means of steam coils, placed in the central column so that the ascending pulp may keep them free from baked mud.

Filtering and Discharging Tailings

Filtering is accomplished by a 250-leaf vacuum-filter plant, constructed on the half-gravity system. That is, filling and emptying of the stock, wash solution, and wash water are effected by gravity through 16-in.

supply pipes, and the various mediums are returned by a centrifugal pump and bucket elevator.

Effluent solutions are sent to precipitation or circulating storage by two 14 by 14 in. Gould duplex wet-vacuum pumps. The final residue is discharged by gravity through the tailings flume to the slime pond.

Precipitation

All pregnant solution to be precipitated is delivered to a 30 by 10 ft. vat, called the press solution vat, from which it is pumped by two 4-in. Krogh chain-driven centrifugal pumps through three Merrill sluice-bar type clarifying presses.

The clarified solution from these presses flows through solution meters to partial-precipitation or complete-precipitation supply tanks.

Only enough solution is completely precipitated to take care of the final washing step at the filters. The rest is precipitated to a value of about 10c. per ton and used as dilution at the upper Dorrr thickeners, for table wash solution and tube-mill feed solution.

Solution to be partly precipitated is pumped by a 10 by 10 in. Aldrich chain-driven triplex pump, zinc dust being added to the pump suction, through a 6-in. pipe line to three 48-frame Merrill triangular presses in the refinery at the head of the mill.

Solution to be completely precipitated is handled in a like manner by one 6 by 7 in. Aldrich triplex pump to one 48-frame Merrill press in the refinery.

Two vats to receive solutions from the presses by gravity are situated at the head of the mill, just below the refinery.

Refining

The precipitation presses are cleaned every 15 days and the product briquetted with the following fluxes (average for 1 year): Borax 2.45 per cent., soda 6.05 per cent., and sand 6.0 per cent.

The melting is done in two double-compartment, carborundum-lined Rockwell furnaces.

During the fiscal year Mar. 1, 1914, to Feb. 28, 1915, we melted 272,-809 lb. of product, which averaged, as it came from the presses, 80.6 per cent. fine in gold and silver.

The following table is an analysis of the completely precipitated product, which has a much higher base content than the partly precipitated product:

Analysis of Completely Precipitated Product

	Per Cent.
Au and Ag.....	74.23
NaCl and KCl.....	1.00
MgO.....	Trace
CaCO ₃	3.60
FeSO ₄	0.26
Al ₂ O ₃	0.60
Pb.....	2.36
Cu.....	2.10
Cd.....	1.00
Zn.....	4.70
Se.....	1.40
SiO ₂	5.50
Al ₂ O ₃	1.40
CaO.....	0.10
Undetermined.....	1.75
	100.00

Insoluble

The analysis of bullion given below is from a representative sample:

Analysis of Bullion

	Per Cent.
Au and Ag.....	93.23
Pb.....	2.41
Cu.....	1.02
Zn.....	0.06
S.....	0.17
Se.....	1.80
Undetermined.....	1.31

100.00

	Assays			Per Cent. of Extraction in Each Department			Per Cent. of Total Extraction in Each Department		
	Au	Ag	Value	Au	Ag	Total	Au	Ag	Total
Battery heads.....	0.32	32.1	\$25.66						
Battery discharge.....	0.21	30.4	22.44						
Extraction in battery.....				34.4	5.3	12.5	34.4	5.3	12.5
Battery discharge.....	0.21	30.4	22.44						
Dorr classifier overflow.....	0.08	27.1	17.86						
Extraction in tube mills.....				62.0	10.8	20.4	40.7	10.28	17.9
Classifier overflow.....	0.08	27.1	17.86						
Table tailings.....	0.06	24.2	15.72						
Extraction by concentration.....				25.0	12.0	8.9	6.0	9.0	8.34
Table tailings.....	0.06	24.2	15.72						
Agitator heads.....	0.05	21.2	13.72						
Extraction lower thickeners.....				16.6	12.4	20.0	3.1	9.33	7.8
Agitator heads.....	0.054	18.9	12.42						
No. 6 agitator.....	0.02	4.85	3.31						
Extraction first six agitators.....				60.0	74.3	73.3	9.4	50.99	40.57
No. 6 agitator.....	0.02	4.85	3.31						
No. 7 agitator.....	0.015	4.2	2.82						
Extraction upper thickeners.....				25.0	13.3	14.8	1.2	2.0	1.9
No. 7 agitator.....	0.015	4.2	2.82						
Filter stock.....	0.014	2.8	1.96						
Extraction second six agitators.....				6.6	33.3	30.5	0.3	4.3	3.3
Filter stock.....	0.014	2.8	1.96						
Filter discharge.....	0.01	2.3	1.58						
Extraction in stock tank and filters.....				28.6	18.0	10.3	1.25	1.56	1.48

* Samples are averages for 30 days, taken from assay records. Silver figured at 80c. per ounce for values.

The extraction from 438,708 tons, up to Mar. 1, 1915, was 96.2 per cent. of the gold, 93.0 per cent. of the silver, and 93.9 per cent. of the money value.

The table at the foot of the preceding page shows the dissolution of both gold and silver through each step of the milling operation, figured both from assay of material as it enters the department and from original assay of ore milled.

General Remarks

The mill has concrete floors throughout. The grinding and concentrating floors drain to the lower Dorr thickeners, and the lower floors to a common sump from which solution from leaks and overflows can be pumped back into mill circuit. The initial saving effected by not laying concrete floors is false economy, I believe, as the waste bound to occur in any cyanide mill will soon exceed the cost of their installation.

The following analysis of the higher-grade ores from the Belmont mine will show the base content, which probably occurs in ores of all grades somewhat in proportion to the gold and silver content:

	Per Cent.
SiO ₂	72.00
Fe.....	2.10
Mn.....	1.60
S.....	2.60
CaO.....	3.10
Al ₂ O ₃	0.42
MgO.....	2.95
Zn.....	3.00
Ag.....	4.88
Au.....	0.06
Pb.....	1.50
Cu.....	1.09
As.....	Trace
Sb.....	0.10
Se.....	0.20
Undetermined.....	4.40
	100.00

The silver occurs as silver sulphide, antimonial silver sulphide, arsenical silver sulphide, with iodine as a base, and native silver; probably somewhat in the following order:

Silver sulphide.....	argentite
Antimonial silver sulphide.....	{ stephanite pyrargyrite
Arsenical silver sulphide.....	{ polybasite proustite
Silver iodide, chloride, bromide, and native silver.	

Tests with ore agitated 48 hr. in fresh solution, without the addition of lead, indicated a cyanide consumption of 2.8 lb., and that 1 lb. of sulphocyanide and 0.02 lb. of ferrocyanide were formed. An extraction of 92 per cent. of the values was obtained. These tests show how quickly fresh solutions become foul after contact with the ore.

In practice lead acetate to the amount of about $\frac{1}{2}$ lb. per ton is introduced to take care of the sulphur, and this tends to form insoluble lead sulphide instead of combining as a sulphocyanide. Along this line experiments have been made with litharge and lead nitrate, but the results were not equal to those obtained with lead acetate. It is figured that to some extent the zinc in the solution from precipitation combines with the sulphur, the same as the lead does. In spite of the addition of lead salts as a desulphurizer, sulphocyanide to the amount of about 2.7 lb. is carried in the mill solutions, and while this cyanide combination does not hinder the dissolution of gold and silver—further than this or any other inert salt tends to increase the viscosity of the solutions—still it is a combination of cyanide that it is almost impossible to break up, and must therefore be considered as a consumption.

Proper alkalinity has a great deal more influence in cyanidation than is generally believed. The principal mission of lime—at least it is so considered in milling practice—is to neutralize the acidity of the ore and to do this at the first opportunity. This would naturally mean adding it to the batteries with the ore feed, but to prevent the coating of the battery screens which this method would cause, it is added as an emulsion to the battery discharge.

Early experiments on Belmont ores indicated that precipitation of the dissolved gold and silver took place during agitation with either a deficiency or an excess of alkalinity. When between 0.8 and 1.5 lb. was carried no precipitation was indicated, and as these limits are satisfactory from a settling and precipitation standpoint, they have been maintained as working limits.

About two-thirds of the total amount of cyanide used is added to the pulp flow at the No. 1 tank in the first series of agitators, and at this point the temperature of the solution is raised as much as possible, usually about 15°. The other one-third is added to the second series of agitators, but instead of "sweetening" the first tank of this series, it is added to the second tank, thereby allowing one tank in which the precipitated solution which is introduced into the second series of Dorr thickeners and between the two steps of the agitation may work, the theory being that the regenerated cyanide in this solution requires no assistance from fresh salts.

Heating the solutions to 90° temperature, in addition to increasing the ultimate extraction about 2 per cent., facilitates the coagulation of the slime, being of benefit in the settling and increasing the rate of percolation in the final filtering.

The following analysis of the circulating solution will give an idea of what salts are carried.

Total solids.....	0.7 per cent.	Per Cent.
KCN.....		0.225
KCNS.....		0.135
CaO.....		0.067
Zn.....		0.045
Cu.....		0.027
SO ₃		0.009
Cl.....		0.015
K ₂ FeCN ₆		0.010

MECHANICAL OPERATION AND COSTS

Crushing and Conveying

The crusher plant has a capacity of about 100 tons per hour and is run only one 8-hr. shift per day. It is driven by one 100-hp. motor on crushers and trommel, one 20-hp. motor on waste conveyor, one 7½-hp. motor on grizzlies and undersize belt, one 20-hp. motor on inclined belt, and one 15-hp. motor on horizontal battery-bin belt and automatic tripper, a total of 162½ hp. The actual power consumption, under full crushing load, is 80 hp.

Concaves for the No. 4 short-head crushers weigh 677 lb., cost \$88, and last two years, crushing about 75,000 tons. These concaves are run about six months, then are set out, rebabbitted, and run for six months longer, at the expiration of which time they are turned end for end and give another 12 months' like service.

Concaves for the No. 7½K crusher last about one year and crush in the neighborhood of 150,000 tons. They weigh 1,120 lb. and cost \$202.

The No. 4 mantle weighs 536 lb., costs \$132, and has a life of two years, handling about 75,000 tons.

The No. 7½ mantle weighs 1,994 lb., costs \$352.65, and lasts 18 months, crushing about 225,000 tons.

The manganese-steel trommel screen weighs 3,776 lb., costs \$572.30, and lasts two years, handling 300,000 tons.

The consumption of wearing parts in the crusher plant is 0.0325 lb. per ton of ore, at a cost of \$0.00555.

Stamping

The 1,250-lb. stamps are made up as follows: 3¾-in. stem 550 lb., tappet 180 lb., boss head 345 lb., shoe 183 lb.

One 60-hp. motor drives 20 stamps with two 6½-in. cam shafts. The stamps drop through 6 in. 104 times per minute. The actual power consumption for each 20 stamps is 52.6 hp.

The 12 mortars are of the narrow type, with 15-in. anvil block base, and weigh 12,000 lb. each. Union Iron Works forged-steel dies $5\frac{1}{2}$ in. high, weighing 159 lb., and chrome-steel shoes 9 in. high, weighing 183 lb., are used. The consumption of die iron per ton of ore stamped is 0.207 lb., and the discarded iron 0.064 lb.; the consumption of shoe iron per ton of ore stamped is 0.179 lb., and the discarded iron 0.108 lb.; making a total of 0.553 lb. per ton of ore, divided as follows: Consumed 0.386 lb., discarded 0.172 lb.

The average service for cam shafts has been 27,214 tons stamped. This life seems short and is mainly due, I think, to close feeding, and to a few shafts of very inferior quality. Still, if close feeding will increase the stamp duty by one-twentieth, the saving in power cost alone will pay for the increased breakage.

Thermit welding of cam shafts has been tried, but without great success. Two shafts were welded, at the following cost per shaft:

Labor.....	\$16.00
85 lb. thermit.....	21.60
Iron punchings.....	0.38
Wax.....	1.75
Gasoline (preheat).....	2.80
Total.....	\$42.53

The first shaft repaired was put into service July 1, 1914, and broke, in the weld, Aug. 8, 1914. The second was put into service Aug. 8, 1914, and broke, in the weld, on Sept. 13, 1914. These two shafts were welded at the break, necessitating their being turned down to original size, which is against the advice of the thermit manufacturers. On Apr. 6, 1915, another shaft was welded from two long, discarded ends, sawed so that the weld would come in the middle box, and a boss $\frac{1}{2}$ in. high and 4 in. wide left, which was taken care of in babbitting the center bearing. This shaft is still in service and may show a life really worth while. Figuring 8.88 tons per stamp day, the 30-day service from the first two shafts welded would cost about 2c. per ton of ore stamped, which, in case of breakage with no spare at hand, would be better than having 10 stamps out of commission for any length of time, and therefore thermit supplies are kept on hand for this and other emergencies.

Traveling on a track $10\frac{1}{2}$ ft. above the tops of the battery posts is a 5-ton crane. Two complete cam shafts, right and left, are assembled on the cam-shaft floor, so that in case of a broken shaft, or even a broken cam which would necessitate the removal of several cams ahead in order to make repairs, the entire shaft is lifted out and a new shaft put in its place, after which the removed shaft is dressed for service and placed on the rack.

The mortar blocks contain 759.35 cu. yd. of concrete. They have a

common base, a cross-section of which is shown in Fig. 4, and are arranged in two 20-stamp blocks and two 10-stamp blocks, the 10-stamp units being on the ends.

Tube Milling

One 100-hp. motor drives two tube mills by double Morse chain drive, a 10-in. chain running from the motor to the line shaft, with a cut-off coupling to one mill and a 6-in. chain with clutch coupling to the second mill. The actual power consumption varies from 98 to 127 hp., for two mills with full load. We are well pleased with this method of transmission. The arrangement is far neater and more compact than the belt drive, and much more efficient. For milling 300,000 tons the cost per ton for chain repairs was \$0.00184.

Our first tube-mill liners were 4 by 4 by 8 in. silex blocks laid flat, in seven mills, and Campbell & Kelly (local foundry) flat white iron in the eighth mill. The average cost of lining with silex was as follows:

Silex per mill, 12,300 lb.....	\$257.68
Cement per mill, 33 sacks.....	35.80
Labor in lining.....	51.56
Labor, miscellaneous.....	50.80
Total.....	\$395.84

This lining gave a life of eight months.

Tons ground per mill.....	15,533
Cost per ton reground.....	\$0.0583
Cost per ton milled.....	\$0.048

In 1913, Komata manganese-steel lining was installed in one mill, at a cost of \$1,784.59 (\$145.30 of which was for labor of installation). This liner ran $16\frac{1}{3}$ months and reground 27,930 tons, at a cost per ton of \$0.064. Then new ribs were installed, at a cost of \$273.04, which should allow 10 months more service, making the cost per ton reground \$0.0457.

The Belmont type of ribbed liner, made by Campbell & Kelly of hard white iron, was installed in the first mill Oct. 4, 1913, at a cost of \$710 in place. At the end of 12 months' run this mill had reground 20,484 tons, at a cost per ton of \$0.0346. At the time this paper is written, after a service of 19 months, it is figured that the entire life of the liner will be two years, making the cost per ton reground \$0.0173.

We have tried French, Newfoundland, Manhattan (local), and Danish pebbles, and have satisfied ourselves that, when power, pebble consumption, and grinding efficiency are considered, Danish pebbles are the most economical.

With Belmont ore the most efficient pebble load was found to be a

little over half full, and the highest grinding efficiency to be with 39 per cent. moisture. (Fig. 5.)

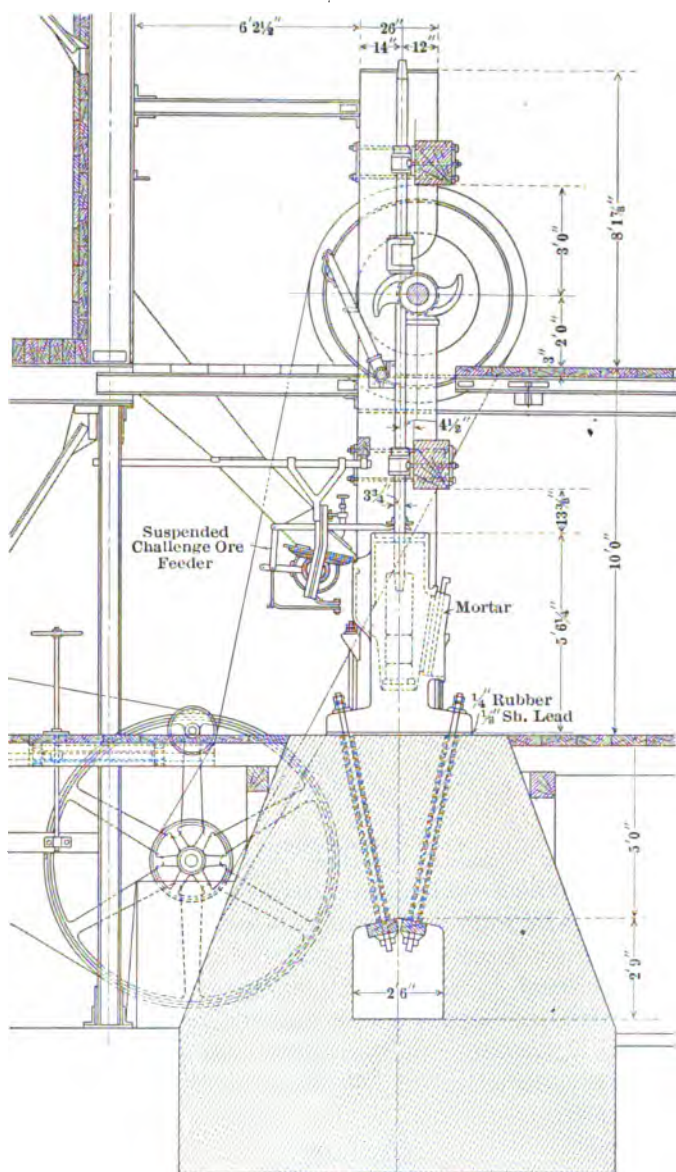


FIG. 4.—SECTION OF MORTAR BLOCK.

We are grinding at this time with seven mills about 75 per cent. through 200 mesh.

Below is given a screen analysis of the product at the different stages of grinding.

Battery Discharge—6-Mesh Tyler Ton-Cap Screens

Mesh.	Per Cent.	Cumulative Per Cent.
+ 10	7.6	7.6
+ 14	12.0	19.6
+ 20	11.1	30.7
+ 28	11.9	42.6
+ 35	8.8	51.4
+ 48	7.8	59.2
+ 65	5.2	64.4
+100	7.3	71.7
+150	6.8	78.5
+200	2.0	80.5
-200	19.5	100.0

Tube-Mill Discharge

+ 35	1.0	1.0
+ 48	3.1	4.1
+ 65	6.5	10.6
+100	20.0	30.6
+150	28.6	59.2
+200	12.7	71.9
-200	28.1	100.0

Dorr Classifier Overflow

+100	1.0	1.0
+150	12.5	13.5
+200	14.2	27.7
-200	72.3	100.0

Concentration

The 16 No. 6 Wilfley tables, running 233 strokes per minute with $\frac{5}{8}$ -in. throw, are driven by one 20-hp. motor (actual power consumption 9.38 hp.).

The tailings launder is situated in a subway under the floor, which subway also carries the wash-solution supply pipe, the water feed to the tables being through a rising pipe with gooseneck discharge into the wash box. This makes a much neater arrangement than having launders and piping above the floor.

Thickening

The eight Dorr thickeners, two of which are equipped with settling trays, are driven by one 10-hp. motor (actual power consumption 5 hp.).

The lower thickeners, which supply feed for agitation, are so arranged that by means of air lifts, which discharge a few inches above the tops of

the thickeners, the pulp is delivered to a small cone-shaped tank which acts as a suction supply for the pump delivering to the agitators. The air for these lifts is regulated by a float in the supply tank, which, in case the flow becomes too great for the pump to handle, closes the air supply until the pulp level is lowered in the tank. With this arrangement no attention whatever has had to be given to the supply of this pump during the three years' operation.

Agitation

Agitation is effected by an air lift through the central column of the agitating tanks. A 6-in. air line is carried about 7 ft. above the agitators and a $\frac{3}{4}$ -in. line dropped to each, inside the center column, discharging down through an open pipe about 18 in. above the bottom of the column. Each tank is 15 ft. diameter by 45 ft. deep and has a capacity of 6,000 cu. ft., requiring $67\frac{1}{2}$ cu. ft. of free air per minute, at 30 lb. pressure, which costs \$0.00003585 per cubic foot, or \$0.0803 per ton agitated. About

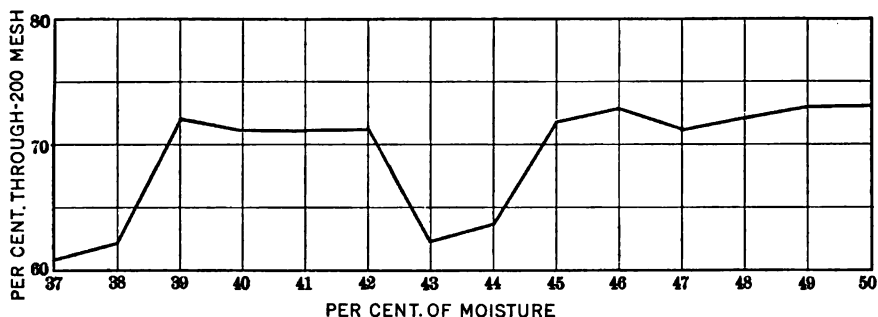


FIG. 5.

300 cu. ft. of air is used in other parts of the mill for air lifts, etc., at a cost of \$0.0485 per ton of ore treated. As previously stated, a better extraction is obtained by heating the solutions, and as the heating efficiency of steam at 5 lb. pressure is only 11 per cent. less than at 125 lb. pressure, our compressor is steam-driven and the exhaust steam used to heat solutions, and, incidentally, these warm solutions heat the mill more evenly, I think, than it could be heated by any other means. Figuring this way, the total cost of heat and air is \$0.1288, as against the cost of air alone from an electrically driven compressor (at \$0.000024 per cubic foot) of \$0.076, making the charge for heating \$0.0528, or less than half of what this charge would be with a double installation. Moreover, we have air that never dies, a very great advantage when the electric power is off, as continuous agitation is insured and settling in the agitators is avoided.

Vacuum is produced by two 14 by 14 in. Gould duplex wet-vacuum pumps, driven by one 20-hp. motor (actual power consumption 11.39

hp.). Connected with this line shaft through a friction clutch is a 15-hp. steam-driven turbine engine, which is used, in case the electric power fails, to maintain vacuum until the charge in process can be finished.

Filtering and Discharging

Filling and emptying at the filters are accomplished by gravity. Solutions and pulp are returned by steel-lined centrifugal pumps and bucket elevator, with a total motor equipment of 60 hp., divided as follows: Stock return elevator 15 hp. (actual power consumption when running 8.17 hp.), stock-return pump 15 hp. (actual power consumption when running 11.2 hp.), wash-solution return pump 15 hp. (actual power consumption when running 9.38 hp.), and wash-water return pump 15 hp. (actual power consumption when running 7.3 hp.).

The following table of costs of elevating wet pulp with both bucket elevator and centrifugal pump may be of interest:

Cost of Elevating Filter Stock Pulp 55 ft. with 20-in. Bucket Elevator

149 ft. of 20-in. Balata belt at \$2.80.....	\$392.70
36 buckets at \$2.90	104.40
220 elevator bolts.....	20.24
7½ lb. washers.....	0.83
41½-in. shaft.....	22.22
21½-in. shaft.....	4.36
Babbitt.....	12.30
Labor to install new belt.....	193.41
Repair labor.....	107.12
Power at \$8.82 hp.-month, 7.3 hp.....	772.56
Chain drive to be installed.....	104.85
Total.....	\$1,734.99
Wet tons elevated.....	1,045,162
Cost per ton.....	\$0.00165

Cost of Elevating Filter Stock 55 ft. with 4-in. Solid-Lined Krogh Pump

Liners.....	\$482.57
Runners.....	126.72
Housings.....	81.00
Bushings.....	36.00
Shafts.....	22.84
Chain and sprocket.....	47.65
Gland.....	1.89
Strand.....	11.25
Labor.....	53.12
Power at \$8.82 per hp.-month, 39 hp.....	408.00
Total.....	\$1,270.68
Wet tons pumped.....	755,255
Cost per ton.....	\$0.00175

Precipitation

Solutions to be clarified are pumped through three Merrill sluice-bar type presses by a 4-in. Krogh chain-driven centrifugal pump, driven by a 10-hp. motor (actual power consumption 5.52 hp.), with a like installation for reserve.

These presses have a maximum capacity of 50 tons each per hour, and actually handle, on an average, 40 tons per hour. They are sluiced 12 times every 24 hr., this operation taking about 6 min. No. 12 Ames-Harris-Neville Co. duck ($9\frac{1}{4}$ to $9\frac{1}{2}$ oz.) is used, and lasts from 70 to 100 days, depending on the amount of suspended matter in solutions to be clarified. For covering one press, 63 yd. of 36-in. duck, costing 27c. per yard, are required, and the labor cost is \$7.

5 presses covered at \$24.50.....	\$122.50
150 days run at 3,000 tons per day, or 450,000 tons clarified.	
Cost per ton clarified.....	0.00027
Cost per ton of ore treated.....	0.0017

Clarified solutions to be partly precipitated, together with the required amount of zinc dust, are fed to the suction of one 10 by 10 in. chain-driven triplex pump and sent through three 48-frame Merrill presses in the refinery at the head of the mill. Solutions to be completely precipitated are handled in the same manner by one 6 by 7 in. triplex pump through one 48-frame press at the refinery.

Two circuits of precipitated solutions are maintained, for economic reasons, as 100 per cent. efficiency in precipitation, as in any other work, necessitates the expenditure of very much more energy and material than is called for by a lower percentage of efficiency, and only enough solution is completely precipitated to take care of the final wash at the filters, and the rest precipitated to from 10c. to 15c. per ton. By this method 1 oz. of bullion was precipitated (average for the past fiscal year) for 0.7808 oz. of zinc dust.

Fiscal year Mar. 1, 1914, to Feb. 28, 1915

171,894 lb. zinc used, or 2,506,214.00 Troy ounces.	
Fine bullion melted	3,209,662.36 Troy ounces.
Zinc consumption 0.7808 oz. per ounce of bullion.	

The precipitated-solution tank below the refinery is fitted with a 16-in. stand pipe, extending from the bottom of the tank to a height of 2 in. above the solution level, and from the lower end of this pipe a 4-in. line leads to the filter wash storage. Into this stand pipe the completely precipitated solution flows, and if not needed for wash at the filters overflows into the tank proper, into which the partly precipitated solution from the refinery flows. This partly precipitated solution is used for dilution

at the upper Dorr thickeners, etc., all that is not needed for this purpose overflowing from the solution tank to the battery storage tank, there joining the circulating solution from the mill, to be used in stamping. By this arrangement only one telltale is required, placed on the battery tank.

Life and Cost of Cloth Covering for Precipitation Presses

Material	Yards per Press	Cost per Yard	Life	Cost per Press	Cost per Ounce Bullion	Cost per Ton Ore Treated
No. 10, 50-in. duck ..	68	\$0.41	6 mo.	\$27.88	\$0.00006	\$0.00123
55-in. sheeting.....	68	\$0.19	1 mo.	\$12.92	\$0.00016	\$0.00342

Refining

The product from the precipitation presses is dumped by hand into steam-jacketed steel cars, run directly under the presses. The loaded cars are switched to one side of the refinery, where they are connected to the steam supply pipe and dried over night to about 15 per cent. moisture. After weighing, and adding the proper fluxes, the product is briquetted in a Grath Little Giant brick machine. Two special dies $3\frac{1}{2}$ in. in diameter are used, making bricks from $2\frac{1}{2}$ to 3 in. thick, depending on pressure and feed, and weighing about 2 lb. each. In practice we are able to briquet 1,000 lb. per hour.

Melting is done in two double-compartment Rockwell furnaces, lined with carborundum, and heated with crude oil and air under about 14 oz. pressure. The ordinary melt of about 150,000 oz. is gotten out in less than 36 hr. after the fires are lighted.

Lining Cost for Rockwell Furnaces. Five Compartments Lined. (Fiscal Year 1914-15)

Labor to line.....	\$80.00
2,765 lb. carborundum screened and reused (labor) ..	21.00
1,500 lb. of carborundum kaolin.....	80.00
10,735 lb. carborundum.....	554.00
4 bbl. water glass	85.00
Total.....	\$740.00
Fine ounces gold and silver melted.....	3,209,662.36
Cost per 1,000 oz.....	\$0.2305

Over the Rockwell furnaces there are steel hoods, connected with a horizontal dust chamber running the full length of the refinery, 80 ft. 3 in. At one end of the refinery, connected with this dust chamber, is a flue, which runs down 20 ft., at an angle of about 45°, to the ground, thence 49 ft. up the hill, at an angle of about 20°, to a 20-ft. vertical stack.

The percentage of gold and silver recovered from this flue, for the past fiscal year, as shown in the following table, was only 0.007 per cent. of the total bullion melted.

Values Recovered from the Refinery Dust Flue

Date	Weight, Lb.	Assay per Ton		Contents Recovered	
		Au	Ag	Os. Au	Os. Ag
4/10/14	50	15.2	1,528.0	0.38	38.2
4/26/14	52	12.0	1,246.0	0.312	32.4
	54	4.8	724.8	0.129	19.5
10/10/14	155	4.8	667.0	0.372	51.6
1/28/15	120	10.0	1,115.6	0.60	66.9
Total.....				1.793	208.6

Fine ounces melted, 3,209,662.36

Recovered from flue, ounces, 210.39

Recovered from flue 0.0067 per cent. of total bullion melted.

From the following table of assays of this dust, taken at different places, it will be seen that the higher values were nearest the hoods and that the dust at the base of the stack was of very low value and small in quantity, which leads us to believe that the loss is inconsiderable.

	Oz. Au	Oz. Ag
Assay from chamber directly over hoods.....	18.8	1,858.2
Assay from base of stack.....	2.6	677.4
Assay general.....	15.2	1,528.0

The costs given below are not the entire refinery costs, and are shown in order to give a better idea of the actual melting costs with Rockwell furnaces.

Labor and Material Costs of Operating Rockwell Melting Furnaces

105,676 oz. melted in 36 hr. from time fires were lighted, in two double-compartment No. 3 Rockwell furnaces.

1,139 gal. oil at \$0.0383	\$43.62
Labor.....	40.95
Power for blower (20 hp.).....	6.00

Total..... \$90.57

Cost per ounce, \$0.0009

Labor segregated:

Head melter, 2 shifts at \$175 per mo.....	\$11.70
Melters, 2½ shifts at \$4.50 per shift.....	11.25
Helpers, 4½ shifts at \$4.00 per shift.....	18.00

\$40.95

The refinery slag is ground in a 6-ft. ball mill and concentrated over a half-size Wilfley table. The concentrates, which carry most of the metallic content, are briquetted with the product for the next melt, and

Milling Costs for the Fiscal Year 1914-15, per Ton of Ore Treated

	Labor	Supplies	Power	Total
Crushing and conveying.....	\$0.036	\$0.020	\$0.010	\$0.068
Stamping.....	0.056	0.155	0.096	0.307
Classifying.....	0.014	0.002	0.006	0.022
Tube milling.....	0.032	0.102	0.230	0.364
Thickening.....	0.024	0.028	0.003	0.055
Concentrating.....	0.028	0.042	0.008	0.078
Agitating slime.....	0.050	0.764	0.007	0.821
Filtering and discharging.....	0.061	0.083	0.027	0.171
Precipitation.....	0.028	0.086	0.020	0.134
Assaying.....	0.009	0.005	0.000	0.014
Refining.....	0.017	0.024	0.002	0.043
Lighting.....	0.001	0.001	0.008	0.010
Shift bosses.....	0.036	0.000	0.000	0.036
Watchman.....	0.012	0.000	0.000	0.012
Surface and plant.....	0.015	0.006	0.000	0.021
Totals.....	\$0.419	\$1.318	\$0.419	\$2.156

Segregation of Supplies Used

	Quantity per Ton	Cost per Ton
Hydrochloric acid.....	0.17 lb.	\$0.0078
Belting.....		0.00336
Cyanide.....	3.52 lb.	0.601
Conveyor parts.....		0.0004
Crusher parts.....		0.0126
Lead acetate.....	0.366 lb.	0.033
Laboratory supplies.....		0.003
Lime.....	5.47 lb.	0.041
Fuel oil.....	2.5 gal.	0.109
Pump parts.....		0.011
Pebbles.....	4.16 lb.	0.057
Tube-mill lining.....		0.023
Zinc.....	0.952 lb.	0.072
Refinery fluxes.....		0.008
Battery shoes.....	0.287 lb.	0.0135
Battery dies.....	0.271 lb.	0.0148
Water (other than mine).....		0.165
Miscellaneous.....		0.14254
Total.....		\$1.318

the tailings from concentration, averaging from \$40 to \$50 per ton in value, are shipped to the smelter. During the first few months of operation these slag tailings were fed again into the mill circuit at the batteries, but a few months of this practice resulted in such an accumulation of copper that it was discontinued in favor of the present method.

The following tabulation shows the rated horsepower, input under full load, and the average monthly consumption of the various motors throughout the mill:

Department and Duty	Rated Horse- power of Motor	Power Consumption while Running		Avge. Hours Run per Day	Monthly Average
		Kilowatts	Horse- power		
Crushing:					
Crushers and trommel.....	100	33.9	45.44	8	15.146
Waste conveyor.....	20	8.5	11.39	8	3.796
Waste-conveyor extension.....	25	14.0	18.76	8	6.253
Grizzly.....	7½	3.0	4.02	8	1.34
Inclined conveyor.....	20	8.0	10.72	8	3.24
Horizontal conveyor.....	15	6.0	8.04	8	2.68
Stamping:					
20 stamps.....	60	39.25	52.6	24	52.6
20 stamps.....	60	39.25	52.6	24	52.6
20 stamps.....	60	39.25	52.6	24	52.6
Classifying:					
8 duplex Dorrs.....	20	6.5	8.71	24	8.71
Tube Milling:					
Nos. 1 and 2 tubes.....	100	87.9	117.8	24	117.8
Nos. 3 and 4 tubes.....	100	95.3	127.7	24	127.7
Nos. 5 and 6 tubes.....	100	82.2	118.0	24	118.0
Nos. 7 and 8 tubes.....	100	73.6	98.67	24	98.67
Concentrating:					
16 No. 6 Wilfleys.....	20	7.0	9.38	24	9.38
Thickening:					
8 Dorr thickeners.....	10	3.73	5.0	24	5.0
Circulating:					
10 by 12 triplex pump.....	30	15.0	20.1	24	20.1
6-in. centrifugal pump.....	30	12.5	16.7	Reserve	
Agitating:					
8 by 10 triplex pump.....	15	6.25	8.37	1	0.697
8 by 10 triplex pump.....	15	6.25	8.37	1	0.697
4-in. centrifugal pump.....	15	9.5	12.75	24	12.75
Lime mixer.....	5	1.0	1.34	24	1.34
Freight elevator.....	5	5.25	7.03	2	0.585
Filtering:					
14 by 14 vacuum pumps (2).....	20	8.5	11.39	24	11.39
6-in. centrifugal pump on Trent agitator.....	15	6.5	8.71	24	8.71
4-in. centrifugal pump, wash-water return.....	15	5.5	7.37	16	4.91

Department and Duty	Rated Horse-power of Motor	Power Consumption while Running		Avg. Hours Run Per Day	Monthly Average
		Kilowatt	Horse-power		
Filtering:					
4-in. centrifugal pump, solution wash return	15	6.08	8.17	14	4.76
4-in. centrifugal pump, stock return.	15	7.5	9.38	10	3.9
Mechanical mixer	10	3.25	4.35	19	3.43
20-in. elevator	15	8.34	11.18	19	8.84
Precipitating:					
10 by 10 triplex pump	30	13.31	17.56	24	17.56
6 by 7 triplex pump	15	5.83	7.81	24	7.81
Zinc feeders	5	2.0	2.68	24	2.68
4-in. clarifying centrifugal pump ..	10	4.12	5.58	24	5.58
4-in. clarifying centrifugal pump for reserve	10	4.12	5.58		
Assaying:					
Crushers	15	5.0	6.7	4	1.116
Hot plates		5.0	6.7	6	1.675
Refining:					
Grinding and concentrating slag and briquetting	15	6.25	8.37	2	0.87
Blower	20	6.91	9.26	2	0.965
Machine Shop:					
General	20	7.5	10.72	12	5.36
Lighting					12.45
Total					813.69

Horsepower per ton milled, 1.627.

Average for year, 1.68.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Copper Deposits of San Cristobal, Santo Domingo

BY THOMAS F. DONNELLY, NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

Introduction

THE Province of San Cristobal is situated on the south side of the island of Santo Domingo about 25 miles west of Santo Domingo city, the capital of the republic. The copper mineralization is found about



FIG. 1.—CAMP BUCARO. THE TOP OF THE RIDGE IN THE BACKGROUND MARKS THE CONTACT OF THE LIMESTONE AND TUFFS.

8 miles north of the town of San Cristobal, in the section known as Bucaro Hill; in San Francisco Hill; on the Nigua River; and on the Jaina River. The district is reached by automobile from Santo Domingo city over an excellent road for 25 miles to San Cristobal, and then, by horseback 6 or 8 miles up the Nigua River to what is locally known as "Camp

Bucaro" (Fig. 1). The trip from the capital takes about 3 hr. The camp is really not more than 10 miles from the sea; yet, since the coast offers no suitable harbor in this vicinity, Santo Domingo city is its only available port.

The district under consideration is situated in what might be called the foothills of the southern watershed of the main Cordilleras, a range



FIG. 2.—LOOKING UP THE NIGUA RIVER AT CAMP BUCARO.

which traverses the center of the island in a slightly north of west direction. The mountainous nature of this section of the country extends almost to the sea, so that at Camp Bucaro the river bed has an elevation of 500 ft. above the sea level. In this vicinity the high peaks rise from 1,200 to 1,500 ft. Up the Nigua River, the mountains become higher and more rugged until, 40 or 50 miles up, where the main ridges are encountered, they attain an elevation of 7,000 ft., with some peaks over 10,000 ft. above tide.

The Nigua Valley is narrow, steep, and rugged. The river itself is small—hardly more than a creek—and shallow enough to be crossed almost anywhere by stepping from stone to stone. However, it has a steep grade, and during the rainy season is often impassable (Fig. 2).

The climate is semi-tropical and very healthy. The thermometer registers around 65° F. during the evenings and night, and rises to about 85° F. at mid-day. There is no regular rainy season in this section. The most rain falls between June and October; but there is rain throughout the year.

General Geology

Briefly speaking, the region consists of a Cretaceous limestone in contact with post-Cretaceous igneous rocks, which are cut in many places by trachyte dikes. These igneous rocks are principally basaltic and basic tuffs¹ and volcanic ashes.² The general direction of the contact is approximately northwest-southeast. The country has suffered violent fracturing and faulting during at least two separate periods. The older series has a general strike of about east and west, and the later system strikes on an average of about N. 30° E. This later system has been productive of the greater movement. Mineralization has taken place in both series of fractures, and each has had its influence on the ore deposition of the district.

The country presents an intricate network of fractures, extending in every direction. On the hills the surface has been greatly weathered, disintegrated, and leached; whereas in the river bed the tuffs are exposed in hard, compact, fresh form, indicating that the atmospheric agencies have exerted a greater influence than the river waters (Fig. 3).

Economic Geology

There are two promising sources of ore in the district: (1) the deposits that should occur at or near the contact of the igneous rocks and the limestone; and (2) the fissure veins.

Of the former the most attractive prospects are found in the north-western quarter of the Bucaro claim on Bucaro Hill (Fig. 4). In this section the following interesting conditions exist:

¹ There seems to be some confusion as to the exact nature of tuffs. They are the fine, fragmental ejections from the explosive eruptions of a volcano. They may afterward be water sorted or cemented to a firm rock. Coarser *ejectamenta* are called volcanic breccias, but in neither do we see much sorting unless produced by subsequent erosion. Tufa is sometimes used in this sense, but the practice is wrong, and should be discouraged. A tufa is a cellular deposit of a mineral spring. It is usually calcareous or siliceous. The term should be confined to this material and not be confused with tuff.

² *Mining and Scientific Press*, vol. xcv, No. 10, p. 305 (Sept. 7, 1907).

1. There is a contact of Cretaceous limestone and igneous rocks—principally tuffs and volcanic ashes—the course of which is approximately northwest-southeast. The limestone is tilted nearly vertical and strikes almost north and south.

2. In igneous the rocks there are numerous stringers of ore and a couple of veins that give promise of productiveness. The general strike of this mineralization is about east and west, and hence will ultimately meet the contact.

3. There is a later system of faults which strike approximately north,



FIG. 3.—UNALTERED TUFF IN THE BED OF THE NIGUA RIVER, SHOWING FRACTURE PLANES IN ALL DIRECTIONS.

and south and dip to the west. This system displaces the former system, and toward the western end of the property seems to have changed the dip of the older series of fractures—near the surface at least—from north to south, or toward the limestone. This later series evidently carries no original copper mineral, and the fissures are generally barren except for some “drag” where they cut the earlier fractures. There are, however, north and south dikes that carry what seems to be original disseminated pyrite, but no copper minerals.

The vein filling consists of quartz, limonite, some specularite, hematite, malachite, chalcopyrite, bornite, and occasionally chalcocite and cuprite. The ore deposition probably took place first in the original



FIG. 4.—BUCARO HILL, FROM SAN FRANCISCO HILL.



FIG. 5.—OUTCROP ON SAN FRANCISCO HILL, SHOWING SHATTERED CONDITION OF COUNTRY ROCK.

fissures, as suggested below; but subsequent disturbances have so shattered the country in general, and the minerals have been leached and re-deposited in the walls to such an extent, that it is now impossible to determine the original walls. This condition may be confined to the surface only, which has been greatly weathered and leached; and the veins may be better defined in depth.

It seems to me that the conditions would be ideal for the formation of deposits along the contact of the limestone and the tuffs in the vicinity of where this series of mineralized fractures meet it—a zone of



FIG. 6.—SAN FRANCISCO HILL, SHOWING OPENING ALONG CROWN COPPER VEIN AT THE TOP OF THE HILL. THE VEIN IS 30 FT. WIDE.

about 4,000 ft. in length. The present development has for its object the exploration of this zone.

The fissure veins are best exemplified on the north side of the Nigua River, in the vicinity of San Francisco Hill. Here they are quite different in appearance from those around Bucaro. The walls are better defined and quartz is much more prominent in the vein filling.

The older fissures on San Francisco Hill strike a little north of east (although this varies somewhat) and dip to the north (Fig. 5). The veins range from fine stringers to a width of 3 or 4 ft. This system of veins has been displaced and offset by a later period of faulting,

with a result that seldom more than 100 or 200 ft. of vein are found in one continuous line. The vein filling is white sugary quartz, carrying as its principal mineral bornite, with some minor quantities of azurite and malachite. The precious-metal values are fairly good, but not uniform.

The more promising veins are the later ones, or those that show a strike of about N. 30° E. The dip of these varies considerably, those on



FIG. 7.—PLATINITA CREEK. OUTCROP NEAR HEAD.

the eastern portion of the property dipping generally to the east, and those on the western end to the west. The vein on which most work has been expended in this region is locally known as the Crown copper vein (Fig. 6). It has been exposed across San Francisco Hill by pits, shafts, and stripping for a distance of about 4,000 ft. This fissure measures in one place 30 ft.; at the Nigua River it is 10 ft. wide. I do not think, however, that it will average nearly this width. The vein appears to have experienced, after its original mineralization, a pronounced fault move-

ment, to which its peculiar character is due. In the hanging wall is found, in places, a layer of rich bornite ore a few inches thick; and filling the rest of the fissure is a talcose gouge, which shows by the presence of rounded nodules of quartz and chalcopyrite covered with slickensides that a great amount of movement has taken place. It is still a question, whether this last-mentioned mineralization took place originally in the fissure, or is a by-product from the older veins, dragged in by the movement. Work is now in progress to determine this.

About the vicinity of Platinita Creek, there are outcrops of rich ore (Fig. 7) with a general strike of N. 30° E. showing indications of strong veins. Hardly any work has been expended on these hitherto but plans have been made for their early exploration.

Genesis of the Deposits

The presence of copper in tuffs is not uncommon. De Launay has observed that the andesitic tuffs at the Boleo mines, near Santa Rosalia, Lower California, have been the probable source of copper for these deposits.³ He concludes that the deposits are due to the sedimentation aided by the vein-forming waters that emerged during the tectonic movement, and that the gold, pyrite, copper, etc., were precipitated and deposited simultaneously with the pebbles of the conglomerate.³ Weed takes exception to this view, and believes that the evidence would indicate that the circulation of mineralizing waters after the sedimentation was the agent of deposition.⁴ Ransome says that in many of the mines north-west of Globe, Ariz., as the Black Warrior, Black Copper, and Geneva mines, the dacite tuffs are richly impregnated with chrysocolla.⁵ J. A. Dresser claims that the chalcopyrite lenses in Quebec are in sheared volcanic rocks that were subsequently changed to schists. He states that during the expulsion of these volcanic rocks copper, gold, silver, etc., were also emitted, to be afterward concentrated by metamorphism and meteoric waters.⁶ The last word on the Rio Tinto deposits in Spain places them also in sheared eruptive rocks that were subsequently changed.⁷

In the above instances the deposits occur either in lenses or veins along more or less defined bedding planes. In the San Cristobal district, the ore is found in fissures that run in all directions.

In the San Cristobal district there are good reasons to believe that the copper, in part at least, has been derived from the tuffs. On the

³ De Launay: *Les Richesses Minérales de l'Afrique*. Translation from the French, *Engineering and Mining Journal*, vol. lxxv, No. 14, p. 519 (Apr. 4, 1903).

⁴ *Copper Mines of the World*, p. 245 (1907).

⁵ *Professional Paper No. 12, U. S. Geological Survey*, pp. 155 to 159 (1903).

⁶ *Journal of the Canadian Mining Institute*, vol. v, pp. 81 to 86 (1902); *Engineering and Mining Journal*, vol. lxxiii, No. 12, p. 412 (Mar. 22, 1902).

⁷ De Launay: *Traité de Métallogénie* (1913).

surface where these igneous rocks are severely weathered and disintegrated, very little or no copper is present; but in the tunnels in Bucaro Hill, it is quite evident that the copper is disseminated through the unaltered tuffs. A slight green stain of copper carbonate may be seen almost anywhere.

The limestone escarpment that runs along the top of Bucaro Hill and denotes the probable contact of the limestone and the tuffs is, in all likelihood, the result of a normal fault. Probably the same forces and movement that caused this produced simultaneously the series of east and west fractures and fissures. This seems to be indicated by the fact that the fissures with this general direction are more numerous and pronounced near the contact, where the forces have been most potent. The dying phase of this activity probably marks the beginning of the vein formation. The atmospheric waters, percolating through the copper-bearing tuffs, met the ascending magmatic waters; and these, rising together in the fissures, deposited their mineral contents in them. The later, or north and south, system of faults, further shattered the district, made the country more pervious to the action of the meteoric waters, and assisted in the lateral segregation. The trachyte dikes that have the same general strike may have come up at this time, and the emanations from these may have been prolific of further mineral deposition. These dikes carry considerable pyrite, but I was unable to find any copper in them in determinable quantities.

Conclusions

One is greatly handicapped in the study of the country: first, because of the dense tropical growth; second, because of the weathering, which has caused a great deal of disintegration and leaching at the surface; and third, on account of the lack of development work, especially underground. I spent a little over four months in the country, and most of this time was devoted to the surveying and mapping of the district preliminary to a more detailed study. I feel, therefore, that the knowledge thus gained is of a superficial nature, but hope that it may be of benefit to subsequent investigators.

I was very favorably impressed by the district. Although still in the "prospect" stage, the mineralization is so abundant and general that great hopes are entertained for the future if operations are properly directed. It will require money to explore and develop the country, and this money will have to be intelligently expended. A large sum has been spent by the various companies operating here, but only a small percentage of it has been used in real exploration work. This has had a tendency to destroy the confidence of investors; but the district itself is not at fault.*

Labor conditions are highly favorable. Native labor is employed, at 50c. per day. Contracts are let to natives at an average of \$1.50 per

* *Engineering and Mining Journal*, vol. xcix, No. 15, p. 641 (Apr. 10, 1915).

running foot for tunnels 5 by 6 ft. in size. Of course the company furnishes all supplies. Native skilled labor can be had at from \$1.50 to \$2 per day. The natives for the most part are pure negroes, and, like all of that race, are great imitators, and, as a consequence, quick to learn. They are a simple, industrious, child-like people, very little addicted to the use of alcohol, and hence easily handled. There are now but few really good miners among them. They would, however, develop skill under some one who could teach them. Like all Southern laborers, they are inclined to be lazy when on day's pay but they work 12 or 15 hr. a day when on contract.

There is an abundance of water handy for mill operations and there is a good site for a power plant on the Jaina River some 40 miles away.

The hillsides are fairly well forested with hard woods. Mine timber, however, is very scarce.

The transportation problem is not as bad as it would at first seem. There are 8 miles of bad road from Bucaro to San Cristobal. This is a crude road, following to a large extent the bed of the Nigua River. There are no bad grades on it, but it is impassable during the seasons when the river is up. The company now operating here has hauled a considerable amount of machinery over this road—some pieces being as heavy as 5 tons—without experiencing any great difficulty. The present mode of transportation by ox teams is, of course, very slow. But a little work would put the road in such condition that a gasoline traction truck could be used over it. From San Cristobal to Santo Domingo city the road is macadamized all the way. It is a government road called the "Carretera" and is always kept in excellent condition. The Clyde Steamship Co. has made a rate of \$4 per ton for carrying concentrates from Santo Domingo city to New York. This rate is, of course, exorbitant, in view of the fact that the rate for shipping ore from Cuba is only 90c. per ton. A little competition in the shipping field, and the active operation of the district, will do much toward lowering this rate to within reasonable bounds.

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A prominent manufacturer of mining machinery not long ago expressed himself as being confident that he was securing the best kind of publicity for his product through the advertising pages of the monthly *Bulletin* of the American Institute of Mining Engineers.

Such an expression coming from such a man is highly gratifying; yet it is justifiable when the high quality of reader and subscriber is taken into consideration. In fact, the majority of those using our advertising pages feel the same. It is to be regretted, of course, that others catering to our field do not realize the means at their disposal for presenting the points of interest of their product to the mining man.

The question has been asked, "Do your members read their *Bulletin*?" They not only read the editorial pages, but they "admit" being up-to-date enough to peruse the advertising section to "see what is new."

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It is safe to say that the traceable returns from advertising in trade publications hardly, if at all, pay for the advertising. The real returns are intangible, and result from the constant publicity, and keeping "everlastingly at it." Who can say what subconscious effect is taking place when an advertisement is "glanced at," to say nothing of the effect produced when the copy is attractive and interests the reader.

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If it has merit, then satisfactory returns are bound to result; but if, on the other hand, the article has no real merit, then it would be the height of folly to expect the publication to do what the manufacturer himself refuses to do.

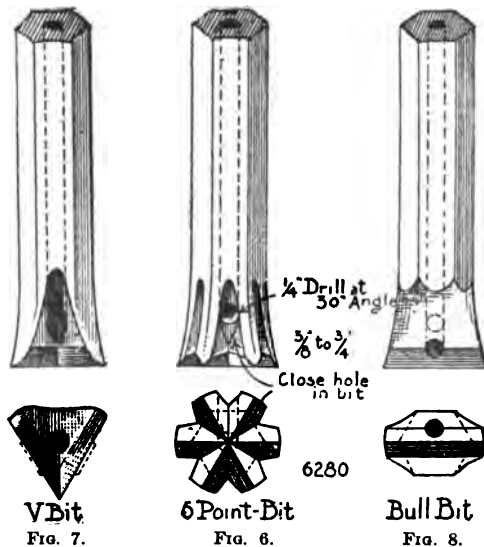
Drill Bits

In our July issue we published data on hammer drills, which we hope have been of interest to the user of machines of this character. Perhaps of more importance than the machine itself is the steel employed for the drill bits.

With each type of drill mentioned hollow steel was used, and we describe herewith some of the special conditions that were met by certain departures from the standard drill-bit practice.

In such work as shaft sinking in loose, caving material, trouble has been experienced from the choking up of the hole in the steel, and the difficulty was overcome by methods as follows: A standard six-point bit was altered as per Fig. 6. The hole in the steel, instead of being carried directly through the center of the bit, was punched through the side about $\frac{5}{8}$ to $\frac{3}{4}$ in. above the cutting edge—the bit itself being forged solid, the hole coming out between the wings.

Clearance of the bit was increased by cutting out the metal between the wings, back to the base of the bit or to the normal diameter of the steel. This reconstruction of the six-point bit permitted it to accommodate itself to the water of condensation and mudding effects. Another bit found to be of great service in almost identical conditions of drilling was a "V" shaped bit, Fig. 7. The cutting edges consisted of two straight



edges, meeting at a point and open at one side similar to a "V," the metal between these cutting edges being cut well back and the hole carried straight down.

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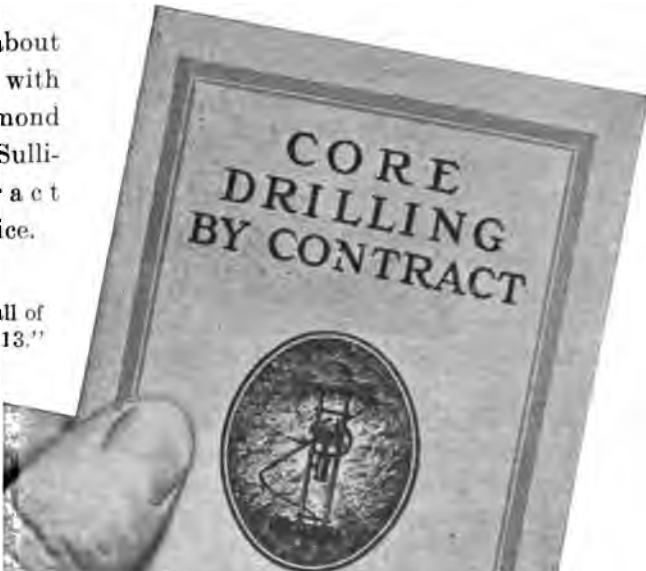
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In sinking a shaft in Colorado through sand rock and blue shale the value of a bit specially created for the job is illustrated. The ground was found to give particularly bad mudding and had a tendency to cement. Various bits were tried, and while it was found that the ordinary form of chisel bit with a single cutting edge gave satisfactory results, further experiments led to a change consisting of carrying the hole in the steel out at the side (Fig. 8), instead of through the end. With multiple cutting edge bits it was found that the steel would jam in the hole, resulting in the loss of many holes and steels, whereas the chisel or flat bull bits could readily be removed and drilled equally as far in this rock.

James A. McIlwee, in commenting on the results obtained by him with Jackhamers in the sinking of a shaft in Utah, remarked that if machines of this class are not mudding properly, the difficulty can often be overcome if the machine and steel be raised occasionally so that the bit is about 6 in. from the bottom of the hole. This will permit the air to blow through the steel and remove the mud from the bottom of the hole. This practice was followed especially in drilling deep holes. He found that the "rose" or six-point bit was a failure in this shaft, whereas a cross four-point bit with liberal clearance between wings gave satisfactory results. The ground in this case was hard quartzite, and it was necessary to temper the steel in cyanide of potassium to protect the corners.

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The De La Vergne Machine Co., of New York, has recently issued Bulletin No. 138, containing some interesting illustrations and descriptions of their type "FH" oil engine for using any crude or fuel oil produced in America.

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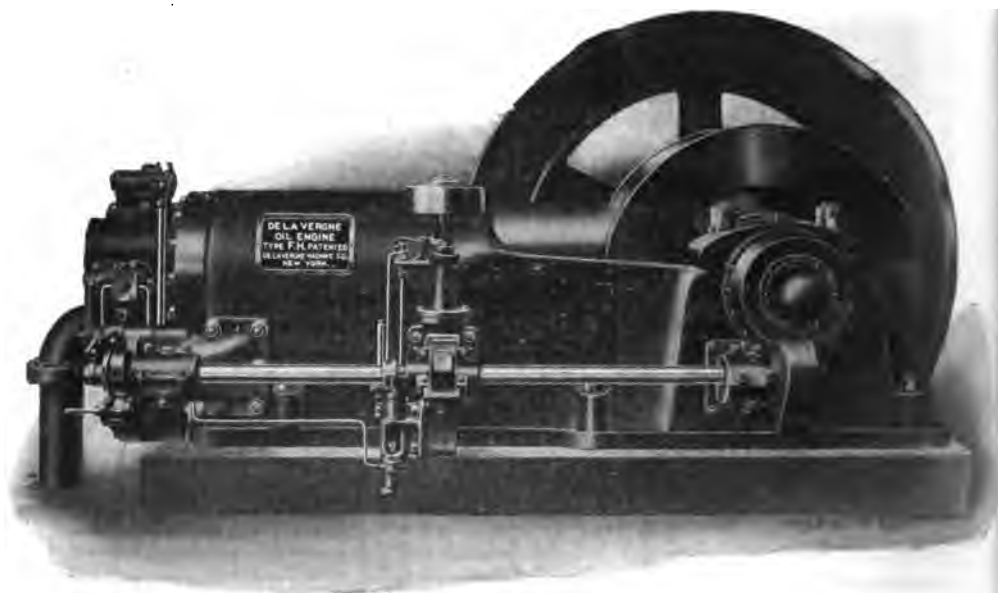
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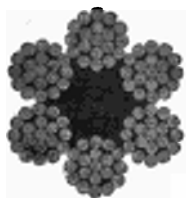
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Gears.—Philadelphia Gear Works, 1124 Vine St., Philadelphia, Pa. A catalogue recently issued by this company is profusely illustrated, and describes the large assortment of beveled gears with planed teeth, rawhide gears, and gear wheels in stock or that may be cut. Data of value to the trade are included. The manufacturer also calls attention to Grant's Treatise on Gears, published by George B. Grant, their engineer, and sent prepaid for \$1. It is a working book on gearing and valuable to the practical man who has to do with gearing problems.

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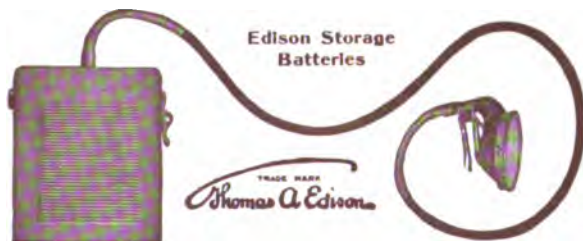
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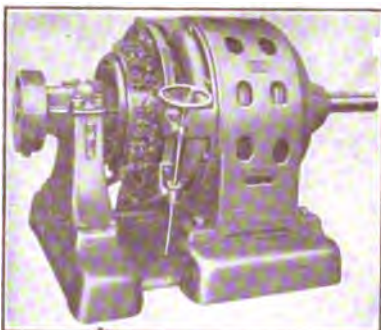
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